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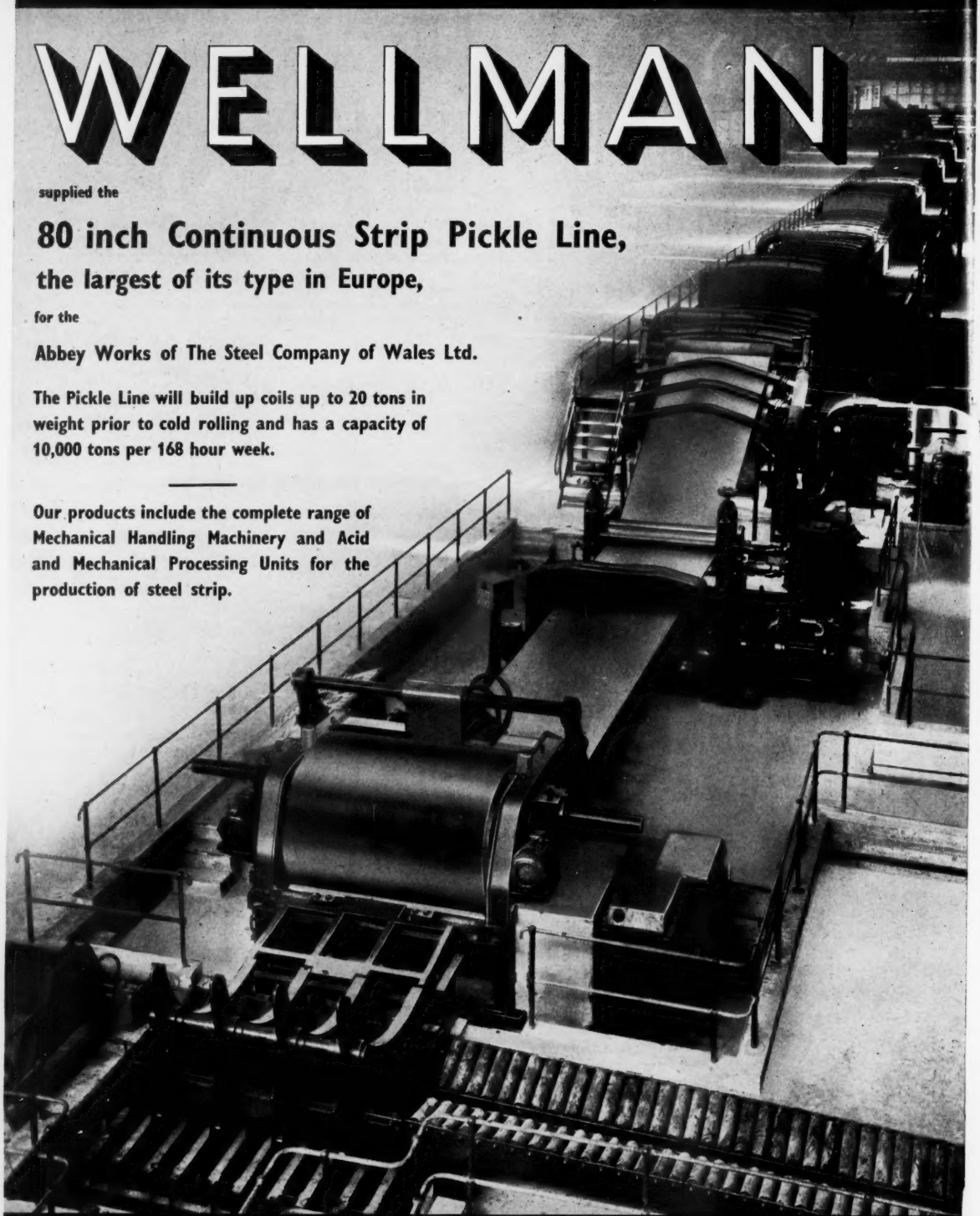

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R. S. BROWN, M.B.E., F.I.M., A.M.C.T.

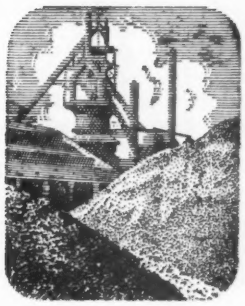
REGINALD S. BROWN was born in Warrington in 1900. He was educated at Warrington Secondary School, Warrington Technical Institute, and the Manchester College of Technology, where he completed part-time courses in Metallurgy and General Chemical Technology, gaining the College Diploma in each course. While taking the course in Metallurgy in 1920, he was awarded the City and Guilds First Prize Silver Medal in iron and steel manufacture, and the Worshipful Company of Ironmongers' First Prize.

Mr. Brown joined Rylands Brothers Limited, Warrington, in 1916. After gaining experience in the sales and general offices, test house, and drawing office, he was attached to the metallurgical laboratories under the late E. A. Atkins. He has since been continuously engaged in research and development in the wire industry, and has contributed papers to The Iron and Steel Institute and to the American Wire Association, who in 1949 presented him with their Medal Award in recognition of his meritorious contribution to the advance of the industry. He was appointed to the Board of Directors of his firm in 1951.

A founder member of the Manchester Metallurgical Society, Mr. Brown served as President for the session 1946-47; he served as President of the Liverpool Metallurgical Society for the session 1952-53. In 1950 he was appointed Chairman of a Committee of Technical Management and Production for the Lancashire Steel Group of Companies. He is a member of the British Iron and Steel Federation Lancashire Area Training Committee and is actively interested in the promotion of technical education in industry. He is a member of the Wire Drawing Committee of the British Iron and Steel Research Association, and was this year appointed Chairman of the Mechanical Working Divisional Panel; he is a member of the Council. He also serves on a number of British Standards Committees. He was awarded the M.B.E. in 1947.

Mr. Brown was admitted as a Fellow of the Institution of Metallurgists in 1948. He was elected a Member of The Iron and Steel Institute in 1932, and, on election to the Council of the Institution of Metallurgists in 1952, was appointed to represent the Institution on the Council of the Institute.

R. S. Brown, M.B.E., F.I.M., A.M.C.T.
Honorary Member of Council, 1952-53



Ironmaking from High-Sinter Burdens

By G. D. Elliot, J. A. Bond, and T. E. Mitchell

SYNOPSIS

Recent work using burdens approaching 100% of sinter in the blast-furnace is described. The results obtained are discussed in relation to heat balances. Comparisons are made with ore practice.

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SINTER has been used at Appleby-Frodingham since 1934. For many years, blast-furnace practice was based on about 33% of sinter in the burden ; it was formerly believed[1] (*loc. cit.*, p. 48) that percentages much in excess of this were uneconomical. The steadily increasing amount of fines arising from changes in the ore bed and from mining practice compelled the use of more than this amount to maintain a balance of raw materials, and further sintering capacity was therefore installed.

The balanced percentage of sinter under today's conditions is approximately 42%. In 1952, however, it was realized that the next few years would see changes compelling the use of about 63% of sinter to maintain a balance of raw materials. Full-scale trials were conducted on the blast-furnaces to assess the effect of these changes. The work was extended experimentally to the use of virtually 100% of sinter on a 25-ft. furnace. The results were so gratifying that, for over a year, Nos. 9 and 10 furnaces, each of 25-ft. hearth dia., have smelted an average of 93% of sinter.

RESULTS

Table I gives the operating results for periods using varying percentages of sinter in the burden. The results of recent months are better than the early results because the operation of the furnaces on high sinter is quite different from operation at lower figures, and experience was needed to make the best of the new practice.

It may be of interest to mention some of the troubles encountered and to compare the new sinter practice with the old, predominantly ore, practice.

Throughout all the work on high-sinter burdens, blowing power was insufficient to determine the maximum blowing rate.

Normally, a blast-furnace is controlled by the operator reading his slags and irons and making adjustments to the burden to maintain iron quality at the desired level. Appleby-Frodingham operators, having self-fluxing burdens, effected these changes by varying the relative proportions of limy and siliceous ores in the burden. When smelting 100% of sinter this control has been mainly passed to the operator of the sintering plant.

The Jurassic ores of Frodingham and South Lincolnshire are extremely variable in composition. The problem of variability is not helped by the ore having to come from many different mines and quarries. Irregularities were so serious that the Robins–Messiter system of blending rough ores was first applied in ironmaking at Appleby-Frodingham in 1939[1] (*loc. cit.*, p. 22 *et seq.*). The maintenance of a uniform analysis on the sinter plant was, and is, a serious problem. It has not yet been solved, but considerable progress has been made.

In the early days of the work being described, variation of sinter analysis frequently gave trouble on the furnaces. At one time, iron was being made regularly with a carbon content as low as 2·75%, as against the previous and present normal of 3·4%. During the same period, iron was leaving the tapping hole at so low a temperature (often as low as 1220° C.) that very serious trouble was encountered because of skulling in hot-metal ladles. These troubles were being encountered when the silicon content of the iron was comparatively high—1% and more. It was formerly the operators' experience that when a blast-furnace lost hearth temperature, it first showed by a fall in silicon content, and only when the silicon had fallen to a low level, say, 0·4%, did the temperature of the iron fall. Iron temperatures were now falling before the silicon. It is believed that these troubles arose from changes in the chemical composition of the sinter.

A typical sinter analysis, which experience has shown to give good furnace operation, is as follows :

	%		%
SiO_2	15·7	Fe_2O_3	35·1
CaO	18·6	FeO	14·7
Al_2O_3	9·3	S	0·21
MgO	1·8	P	0·62
MnO	1·1	Fe	36·0

Manuscript received 10th August, 1953.

Mr. Elliot, Mr. Bond, and Mr. Mitchell are at the Appleby-Frodingham Steel Company, Scunthorpe, Lincs.

If sinter is regarded as a pre-smelted slag, which it largely is, the diagrams[2–5] show that the desirable sinter composition for good ironmaking, under Appleby-Frodingham conditions, lies in a field of low melting points. A change to a more siliceous sinter, or to a more limy sinter, increases the melting point of the slag. With varying analyses of the sinter, the furnace has to work with a constantly varying slag composition ranging from too limy to too acid.

As will be shown later, sinter relieves the furnace of much of the work normally carried on in the stack, so that there is no longer the same reserve of heat in the process. It is believed that sinter does not take much part in furnace reactions until a long way down the furnace, so that any change is felt very rapidly in the hearth. Whatever may be the underlying theory, whether silica is easier to reduce from sinter than from ore, or whether it is simply changes in melting point or viscosity, the fact remains that, as the sinter-plant operators became increasingly able to maintain a steady analysis, all these furnace troubles disappeared.

PECULIARITIES OF ORE-BURDEN PRACTICE

Lean-ore practice, with its attendant high slag volume, has features not found in normal rich-ore practice. The following are the principal peculiarities of the former ore practice at Appleby-Frodingham :

(1) Inability to carry high blast temperatures—a fair average was 450° C. Higher temperatures have always produced hanging of the stock column.

(2) Inability to blow at high pressures. It has long been recognized that with a sticking furnace it was usually possible to get in more wind at, say, 15–16 lb./sq. in. than at 22–24 lb./sq. in. This feature cannot be dismissed as inefficient operation. It was commonly supposed to be a difficulty associated with high slag volumes and possibly with the low bulk density and easy breakdown of Jurassic ores.

(3) Inability to maintain blast volumes through a period of stock hanging. At Appleby-Frodingham the most effective way of restoring uniform stock movement and of getting back to normal operating rates was to cut blast volume during periods of hanging. If the normal blast pressure was maintained or exceeded, the furnace invariably became tighter and tighter so that less and less wind was blown until a reduction in pressure was enforced.

(4) Inability to produce anything approaching the same iron output as rich-ore practice in similar-sized stacks. Again, a high slag volume was believed to be such a drag on the furnace that it was impossible to make a high tonnage of iron.

(5) Inability to achieve anything approaching as low a coke consumption per ton of pig as in rich-ore practice. Here, the operators felt themselves to be on very firm ground. Applying all the usual corrections for slag volume, a coke consumption of about 22 cwt., with a slag volume of 25 cwt., compared favourably with a coke consumption of, say, 17 cwt. and a slag volume of 12 cwt.

(6) Inability to burn as much coke in unit time as is possible in rich-ore practice in furnaces of similar size.

Table I

COMPARISON OF PRACTICE USING VARYING PROPORTIONS OF SINTER

(Twin furnaces, Nos. 9 and 10, 25-ft. hearth dia.)

	1	2	3	4	5	6*	7
Nominal sinter, %	35	42	63	95	93	94	Nil
4-week period ending:	27.10.51	27.12.51	12.4.52	5.7.52	6.6.53	4.7.53	28.3.53
Iron made, tons	3,497	3,620	4,460	4,510	5,170	6,090	4,140
Burden, cwt./ton	63·67	62·07	60·75	54·46	55·91	52·99	71·84
Coke, cwt./ton	22·66	22·02	19·54	17·51	17·97	17·80	23·70
Blast pressure (at furnace), lb./sq. in.	15·20	15·90	19·30	17·90	18·70	19·30	14·30
Blast temperature, ° C.	438	491	562	620	626	622	507
Top-gas temperature, ° C.	203	185	207	225	241	260	143
Northants—dry rubble, %	30·20	28·70	16·00	2·02	0·72	1·60	50·57
Local—double-screened and bedded, %	34·10	29·30	20·00	2·97	3·06	—	48·40
Actual sinter, %	35·70	42·00	63·50	95·01	92·74	94·13	—
Scrap, scale, etc.	—	—	0·50	—	2·71	3·41	1·03
Manganese ore (experimental)	—	—	—	—	0·77	0·86	—
Metallic iron in burden, %	30·26	31·66	32·85	36·03	35·78	36·65	27·30
Flue dust, cwt./ton	1·79	1·57	0·74	0·45	1·32	1·76	0·63
Slag volume, cwt./ton	23·90	23·90	25·60	25·70	25·20	25·30	22·40
Slags:							
SiO_2, %	33·35	34·35	34·40	34·80	34·47	34·60	34·20
Al_2O_3, %	20·16	19·90	20·04	19·97	19·10	20·10	19·75
MnO, %	1·19	1·31	1·35	1·41	2·04	2·06	0·97
CaO, %	39·40	38·90	38·20	37·91	37·77	37·00	38·80
FeO, %	0·64	0·62	0·70	0·86	0·85	0·84	0·64
Fe, %	1·51	1·34	1·60	1·57	1·45	1·70	1·55
MgO, %	3·68	3·70	3·89	3·74	3·60	3·68	3·82
S, %	1·83	1·64	1·50	1·27	1·28	1·36	2·12
CaO/SiO_2 ratio	1·18	1·13	1·11	1·09	1·09	1·07	1·13
Carbon used in furnace, cwt./ton of pig	19·71	19·16	17·00	15·22	15·63	15·49	20·61
Carbon used in sinter plant (including ignition gas equated at valve), cwt./ton of pig	2·62	2·97	3·95	5·74	4·85	5·10	—
Total carbon consumption, cwt./ton of pig	22·33	22·13	20·95	20·96	20·48	20·59	20·61†

* Results for one week only, indicating future possibilities.
† This indicates the effect of a fully prepared ore burden. Such a practice is economically unsound, as all fines arising in preparation have been discarded.

Slag volume, varying raw materials, the inability to blow at high pressure, and many other factors were given as reasons for this shortcoming.

(7) Ore practice was associated with comparatively low top-gas temperature. This was attributed to the cooling effect of the high amount of water and CO_2 charged into the furnace. It has not yet been necessary to add water to high-sinter charges to keep the top-gas temperature low, as is commonly practised with rich ores.

Each of the above items has prevailed in Appleby-Frodingham practice for many generations, and none can be dismissed as the whim of individual operators. Results now being obtained when smelting 100% of sinter show that all these factors have altered and must be reconsidered in the light of recent knowledge.

Control of sinter analysis is not yet sufficiently close for 100% of sinter to be carried all the time. It is frequently necessary to adjust slag basicity by adding up to 5% of siliceous or limy ore, so that, in practice, the average sinter carried is approximately 95%.

When the foregoing factors are compared critically with sinter practice, some light is shed on the theory of the blast-furnace.

HIGH-SINTER PRACTICE

Blast Temperature

It is now possible with high-sinter burdens to operate smoothly with blast temperatures of 750° C. This is not regular practice, because of the necessity to maintain a reserve of heat to meet changes in sinter composition ; but when sinter can be made to a uniform analysis, regular blast temperatures at about this figure will be possible.

Table II

EXTRACTS FROM COMPARATIVE PRACTICE TABLES

	35% Sinter	42% Sinter	63% Sinter	95% Sinter
Hot-metal production, tons/day:				
Actual	500	517	637	645
Adjusted for delays	514	558	640	667
Adjusted hot-metal production:				
Tons/day/sq. ft. of hearth area	1·04	1·14	1·30	1·36
Tons/day/100 cu. ft. of working volume	1·11	1·40	1·61	1·67
Net coke consumption rate, lb./ton of hot metal	2,538	2,466	2,188	1,961
Net carbon rate	2,074	1,999	1,784	1,604
Flue dust and sludge recovered, lb./ton of hot metal	200	176	83	50
Average silicon in hot metal, %	1·35	1·36	1·12	0·95
Average sulphur in hot metal, %	0·081	0·100	0·117	0·101
Slag analysis (%):				
Bases/acids	0·81	0·79	0·77	0·76
Bases/silica	1·29	1·24	1·22	1·20
Calculated slag volume, lb./ton of hot metal	2,677	2,677	2,979	2,990
Air-blast rate at furnace (30 in. Hg, 60° F.):				
Total cu. ft./min.	46,400	47,200·	47,200	43,700
Cu. ft./min./sq. ft. of hearth area	94·5	96·0	96·0	89·0
Cu. ft. of blast air/lb. of carbon in coke	62·5	61·0	59·5	58·9
Blast temperature, ° F.	820	915	1,045	1,150
Dry top gases, cu. ft./ton of iron	173,600	165,200	144,480	126,000
Ratio CO/CO_2	3·08	2·85	3·03	3·03
Gross calorific value, B.Th.U./cu. ft.	101·2	102·2	103·2	100·7
Gross calorific value, B.Th.U./ton of iron	17,568,320	16,882,880	14,910,560	12,688,480
Sensible heat in top gas, B.Th.U./ton of iron	1,537,760	1,323,840	1,249,920	1,133,440
Theoretical heat generated in furnace, B.Th.U. × 10^6:				
From gasification of C to CO	6·888	6·496	5·824	5·102
From gasification of C to CO_2	5·986	6·272	5·454	5·314
Plus heat from hot blast	2·066	2·173	2·178	2·162
Total heat input, B.Th.U. × 10^6	14·941	14·941	13·457	12·578
Less sensible heat in top gas, B.Th.U. × 10^6	1·538	1·324	1·250	1·133
Net heat absorbed in process, B.Th.U. × 10^6	13·403	13·617	12·207	11·444
Equivalent process heat/lb. of dry top gas, B.Th.U.	1,018	1,090	1,119	1,188
Total heat lost, B.Th.U./lb. of fuel carbon gasified	9,650	9,550	9,500	9,070
Less heat in blast, B.Th.U./lb. of fuel carbon gasified	1,045	1,140	1,280	1,417
Net heat lost in top gas, B.Th.U./lb. of fuel carbon	8,605	8,410	8,220	7,653
Net heat utilized in furnace, B.Th.U./lb. of fuel carbon	5,945	6,140	6,330	6,897
Net heat utilized in furnace, % of available heat	40·8	42·2	43·5	47·4
Fuel carbon gasified in fce., lb./ton of hot metal	1,978	1,905	1,702	1,525
Carbon burned at tuyeres, lb./ton of hot metal	1,773	1,665	1,452	1,290
Carbon gasified at tuyeres, %	89·8	87·4	85·2	84·6
Heat required to dissociate blast moisture at 5774 B.Th.U./lb., B.Th.U.	354,000	260,000	226,000	326,000
Development of heat at tuyeres, 10^6 × B.Th.U./ton Hot metal:				
Heat from carbon gasified to CO	7·806	7·336	6·384	5·690
Plus heat in blast	2·066	2·173	2·178	2·162
Less heat required for dissociation of blast moisture	0·396	0·291	0·253	0 365
Net total heat at tuyeres, B.Th.U. × 10^6	9·476	9·218	8·309	7·486
% of hearth heat in hot blast	21·8	23·6	26·2	28·9

Blast temperature has always been a mysterious phenomenon. In lean-ore, high slag-volume practice, a too high blast temperature leads to sticking and hanging. For many years, reduction of blast temperature has been a common way of dealing with a hanging furnace.

A colder blast means a less concentrated zone of combustion and therefore an upward extension of high temperature, which brings about the melting of incipiently fused material, either on the bosh wall or in the stock itself. This is necessary because of the wide melting range of Frodingham and Northants ores (approx. 700–1200° C.). In other words, the liquefaction of ores, etc., is occurring over a much greater furnace height than if the ore had an instantaneous melting point. A very large volume of gas is necessary to deal with these conditions, because contact time is so short. Hence, it is necessary to burn much more coke in the hearth than is needed for hearth reactions, and the resulting volume of gas is necessary to carry out stack reactions. Especially is this so when it is remembered that CO_2 from the breakdown of carbonates will tend to slow down the reaction speed of the gas. This slowing-down is offset by supplying a greater mass of the reacting gas.

This is true so long as much ore is being charged. It is necessary to burn more coke then is needed for hearth requirements and to use lower blast temperatures to extend the gradient.

Now consider the case for 100% of sinter. There are virtually no volatiles to drive off, so the same amount of sensible heat is not required in the stack. Hence, the gas can be at a lower temperature and/or there can be less gas volume. In addition, sinter has a very narrow melting range (1020–1110° C.) so there is no longer the need to flatten the temperature gradient. The total heat requirements of the process are decreased because of the reduced thermal load in the stack. Expressed in another way, as it is no longer necessary to transmit a large amount of sensible heat to the stack, heat supplied to the hearth as sensible blast heat can be used there.

Table II shows that the amount of top gas per ton of pig falls from 173,600 cu. ft. to 126,000 cu. ft. in changing from 35% to 95% of sinter.

Blast Pressure

It is now possible, when smelting sinter, to operate the furnace at 23 lb./sq. in., which is the limit of the existing blowing plant (new blowers are under construction). All recent experience indicates that blowing pressures appreciably higher than 23 lb./sq. in. are now possible, but this is a considerable increase on the former limit of about 17 lb./sq. in. Formerly, if

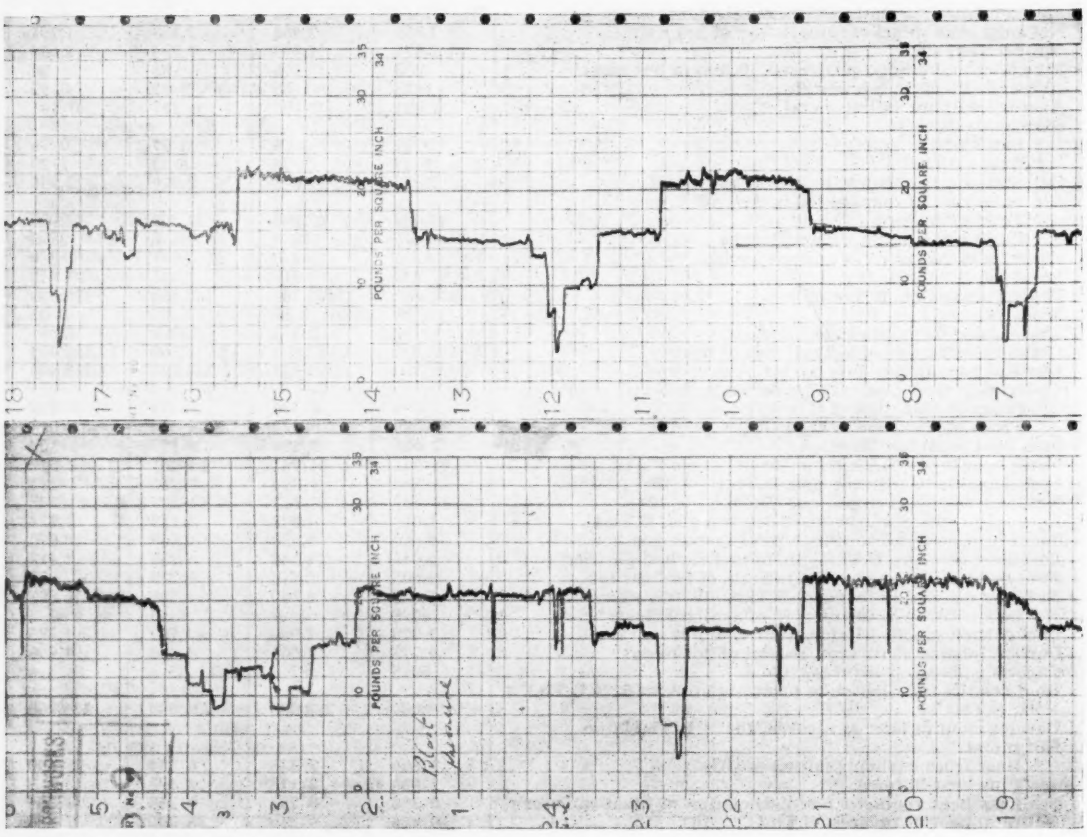

Fig. 1—Typical blast-pressure chart for No. 9 furnace (1st June, 1953)

blast pressure was increased above 17 lb./sq. in., the stock column tightened so that after a very short time less air was being blown at the higher pressure than at the previous lower pressure. This was believed to be due to lifting velocities in the stack being replaced by carrying velocities so that fine ore particles were carried with the gas stream and sealed the voidage formerly existing above. This, of course, was aggravated by the presence of fine ore, or of sinter containing unsintered material, in the burden as charged. The theory was strengthened by the fact that the ores were known to break down to powder at comparatively low temperatures and were of much lower bulk density than most ores smelted in the world today. These factors should not now apply, because sinter does not break down in the same way. In addition, sinter gives more voidage to the stock column, both because of its size and shape and because of its inherent macro-porosity. Also, laboratory tests indicate that sinter is comparatively immune from carbon deposition. Thus, the sinter charged at the stockline descends to its point of melting, or smelting, virtually unchanged.

The subject of blast pressure cannot be discussed at length until experience has been obtained with blowing at much higher pressures. This should be possible within the next few months.

Maintaining Blast Volume during Periods of Stock Hanging

Much of what was said under the previous heading also applies here. Hanging has to be more serious in rich-ore than in lean-ore practice before it is necessary to cut the blast volume. On ore burdens at Appleby-Frodingham, persistent hanging always results in a reduction in blast volume, but on high-sinter burdens the same volume cuts are not required to restore stock descent. This change is believed to be due to reduced carbon deposition, narrower softening range of the raw materials, and better size distribution of the charge.

Iron Output

As most of the work applies to 25-ft. furnaces, figures are discussed only for furnaces of this size, although much check work has been done on 18-ft. furnaces. No. 10 furnace was blown in as a 25-ft. furnace in September, 1951, but 100% of sinter was not smelted until June, 1952. No. 9 furnace was blown in with a 25-ft. hearth in October, 1952, and immediately went on to high-sinter burdens.

Discussion of outputs is necessarily restricted because ample blowing power is not yet available. The blowers were installed in 1939 when present outputs were not visualized. The blowing plant consists of three gas blowers and all three are used, except when maintenance (which is high) necessitates a shut-down. The third engine is swung from furnace to furnace at approximately 5-hr. periods to level out production from the two furnaces. Figure 1 is a typical pressure chart showing this operation. This type of practice would be quite impossible under former conditions. It is a significant pointer to what can be achieved in the future. Figure 1 is not intended to represent what the operators believe to be good practice.

Even with blowing power restricted to 23 lb./sq. in., a production of over 900 tons/day has been possible on many days. Outputs of over 6000 tons in a week have been obtained. This production, together with 1150 tons of slag per day, is much more satisfactory than was visualized by the operators. Outputs of 1000 tons per day should be obtained when greater

Table III

HEAT BALANCES OF COMPARATIVE PRACTICES

	35% Sinter		42% Sinter		63% Sinter		95% Sinter	
Heat generated:	B.Th.U.	%	B.Th.U.	%	B.Th.U.	%	B.Th.U.	%
From gasification of C to CO	6,880,000	46·1	6,500,000	43·5	5,820,000	43·3	5,101,000	40·6
From gasification of C to CO_2	5,980,000	40·0	6,270,000	42·0	5,450,000	40·5	5,320,000	42·2
Heat from hot blast	2,070,000	13·9	2,170,000	14·5	2,180,000	16·2	2,162,000	17·2
Total heat input	14,930,000		14,940,000		13,450,000		12,583,000	
Heat expended:								
For metal:								
Reduction of iron oxides, etc.	6,920,000		6,900,000		6,800,000		6,736,000	
Decomposition of calcium phosphate	163,000		153,000		152,000		147,000	
Sensible heat for iron	1,106,000		1,106,000		1,106,000		1,106,000	
Total for iron	8,189,000	58·0	8,159,000	58·3	8,058,000	60·4	7,989,000	65·0
Total for slag (sensible heat)	1,925,000	13·7	1,925,000	13·8	2,150,000	16·2	2,163,000	17·5
For evolution of volatile matter:								
Evolution of CO_2 from burden	657,000		582,000		412,000		129,000	
Evolution of combined water from burden	424,000		387,000		254,000		71,000	
Water dissociated to make up H_2 in gas	969,000		1,321,000		939,000		439,000	
Sensible heat of moisture in gas	194,000		150,000		120,000		60,000	
Total for volatiles	2,244,000	15·9	2,440,000	17·4	1,725,300	13·0	699,000	5·7
Sensible heat of dry gas	1,345,000	9·6	1,175,000	8·4	1,130,000	8·5	1,075,000	8·7
Decomposition of blast moisture	397,000	2·8	290,000	2·1	253,000	1·9	365,000	3·1
Total heat expended	14,100,000		13,989,000		13,316,000		12,291,000	
Total heat generated (as above)	14,930,000		14,940,000		13,450,000		12,583,000	
Balance (heat loss by radiation, etc.)	830,000		951,000		134,000		292,000	

blowing power is available. The production of these furnaces would then compare favourably with the majority of furnaces of this size, irrespective of the raw materials used.

Coke Consumption

The coke consumptions on high-sinter practice have averaged rather less than 18 cwt./ton of pig. This figure can be reduced by carrying higher blast temperatures, but, as mentioned earlier, these are not yet possible because of sinter variations. However, coke consumptions of 17½ cwt. have been maintained for several weeks together.

In Britain the average coke consumption has been given[6] at 20·30 cwt., with an average slag volume of 17 cwt., each per ton of pig. In America, the average coke consumption was 18·44 cwt., with a slag volume of 11·5 cwt.

The results obtained from sinter burdens on Nos. 9 and 10 furnaces compare very favourably with these figures, especially as the slag volume is 25 cwt./ton of pig. This, of course, does not represent the total carbon consumption in making a ton of pig iron; allowances must be made for the carbon used in the manufacture of sinter, and for ignition gas. Such corrections have been made in Table I, from which three important points emerge :

(1) The overall carbon consumption per ton of iron is decreased by making the iron *via* sinter.

(2) As the percentage of sinter in the burden increases, an increasing amount of the carbon demand is satisfied by using comparatively low grade fuel, such as coke breeze. Indeed, the most important contribution that sinter can make to the overall economy may well be to conserve the coking-coal resources of the country.

(3) The actual increase in carbon utilization is even greater than the figures indicate, because the sensible heat of the sinter is wasted. With a 95% sinter burden, the equivalent of the sensible heat lost in cooling the hot sinter as discharged from the machine is almost 2 cwt. of coke per ton of iron.

The thermal balance sheets (Table III) for furnace operation over periods of one month at varying ranges of sinter, demonstrate clearly where the coke savings arise. The main savings are, of course, in dealing with volatiles, and it may not be an over-simplification to say that it is more efficient to use the sintering plant as a lime burner than to use a blast-furnace for this purpose. Carbon used in the sinter plant is burnt wholly to CO_2 with an evolution of approximately 14,000 B.Th.U./lb. of carbon. In the furnace, carbon is burnt to a mixture of CO and CO_2 with an evolution of approximately 7000 B.Th.U./lb. If this is correct, high-sinter burdens should give important savings in any kind of blast-furnace practice when calcium carbonate enters the blast-furnace.

Tigerschiöld[7] has shown that lime-bearing sinter resulted in a saving of about 110 lb. of coke per ton of pig iron. In no published work on sinter smelting in Sweden are coke savings attributed to the reduction in the volatile load in the blast-furnace. The removal of CO_2 from an average limestone charge of 8 cwt./ton of pig demands the equivalent of about 120 lb./ton of coke. The similarity of this figure to that quoted by Tigerschiöld should be noted.

Tigerschiöld[8] has also said : ". . . lime additions to sinter are more desirable when the material to be sintered contains 8% to 10% silica than if it contains only 5% silica," *i.e.*, the savings are greater when a larger amount of calcium carbonate is sintered than when a small amount is sintered.

Attention is also drawn to the increasing percentage of the heat requirements satisfied by the sensible heat of the blast.

Tonnage of Coke Burnt

Here, again, the argument must remain unproven until ample blowing power is available. For many years, operation with lower-sinter burdens was impossible (because of the factors already listed) at anything more than about 70% Rice rating.[9] (This much-criticized yardstick is still believed by Appleby-Frodingham operators to be a reasonable measure of performance.) With high-sinter burdens, and utilizing existing blowing power to the limit, Rice ratings of 85% on the 25-ft. furnaces have been obtained. On the 18-ft. furnaces, Rice ratings of up to 93% have

Table IV

SULPHUR INPUT ON Nos. 9 AND 10 FURNACES

Year	Sulphur Input, lb./ton of pig	Sinter Used, %	Sulphur in Slag, %	CaO/SiO_2	Slag Volume, cwt./ton	Coke Rate, cwt./ton
1941	42	32·5	1·42	1·13	24·2	21·62
1942	44	32·9	1·47	1·16	24·2	21·45
1943	48	32·1	1·65	1·23	24·0	22·01
1944*	55	27·7	1·70	1·29	28·3	24·96
1945	52	23·0	1·81	1·26	24·0	22·64
1946	52	21·4	1·82	1·29	24·0	23·71
1947	53	24·6	1·86	1·31	24·1	24·05
1951	48	35·7	1·83	1·18	23·9	22·66
1952	49	42·0	1·64	1·13	23·9	22·02
1952	46	63·5	1·50	1·11	25·6	19·54
1952	39	95·0	1·27	1·09	25·7	17·51
1953	38	96·0	1·28	1·04	25·2	17·88
1953†	55	—	2·12	1·18	22·4	23·70

* For wartime reasons it was necessary to charge sand, resulting in a high slag volume.
† Ore used was 52% Dried Northants and 48% Frodingham, crushed at 3 in. and screened at ¾ in.

Note—Because only fines are being sintered, and as slag-making constituents tend to segregate to fines, slag volumes increase as sinter is increased.

been achieved with only about 75–80% of sinter in the burden. If sinter is made so that 100% can be used in a 25-ft. furnace, that sinter is too siliceous for 100% to be used in an 18-ft. furnace. For obvious reasons, it is desirable to cater for the larger furnaces. It is confidently believed that Rice ratings of 100% are possible when smelting 100% of sinter, and when sufficient air is available at a high enough pressure.

Top-Gas Temperature

With the change to a high-sinter burden, the operators believed that water would have to be added to control the expected higher-temperature top gas. However, water has not been required, even at much higher rates of operation. This is most probably because the work carried out in the furnace stack—and therefore the amount of sensible heat transferred to the stack—has been considerably reduced.

It is interesting to speculate by how much the furnace height could be reduced if designed to deal with wholly sinter burdens. Swedish experience in this direction is worthy of close study.

SULPHUR

Sulphur is increasingly a problem at Appleby-Frodingham. Table IV shows the sulphur input to Nos. 9 and 10 furnaces over some years. Sinter burdening means a considerable reduction in the sulphur load for two reasons :

(*a*) About 70% of the sulphur going to the sintering plant is removed in the sintering process.

(*b*) The reductions in coke consumption in the blast-furnace result in a further and larger reduction in sulphur going to the blast-furnace.

The two factors together mean that the same grade of iron can be manufactured from a less basic slag with a lower demand for coke (*see* Table IV).

Table IV is interesting in relation to the finding of Swedish operators, who have shown that the use of 100% of sinter has reduced the total amount of limestone needed per ton of pig iron.[7] When fluxing lime is charged to the furnace as $CaCO_3$, it cannot become effective, because of the high melting point of CaO, until very low in the furnace. When $CaCO_3$ goes through the sintering process, it rarely exists in the sinter as free CaO. Sintering is a melting process and the CaO is absorbed as a slag constituent with a much lower melting point than CaO. That being so, in the blast-furnace, sintered limestone probably comes into action (especially as a desulphurizing agent) at a much lower temperature, and therefore a higher level, than free CaO. It is believed that one of the principal advantages of using a high-sinter burden is the fact that sinter—in these proportions—is essentially a pre-formed slag.

CONCLUSION

The foregoing notes attempt to mention some of the important effects of changing from a burden based wholly on home ores and containing about 35% sinter to a burden consisting of 95% of sinter or more. Because of the comparatively short time occupied by the work and because of limitations of blowing power, this must be regarded as an interim report. The operators are convinced that the future will see an increasing use of wholly sinter burdens. It is suggested that the practice has much to commend it, for both lean- and rich-ore working. For many years, the majority of pig iron in Sweden has been produced from wholly sinter burdens—the fluxing lime being introduced *via* the sinter—and for many years the coke consumption at many Swedish furnaces has been enviably low. Appleby-Frodingham experience confirms that Swedish experience applies equally to large (25-ft.) furnaces using low-grade burdens.

Logically, the use of 100% sinter burdens must be correct. It extends the trend of many years in relieving the furnace of as many duties as possible, so giving it more time to do its primary job of reducing iron oxides to iron.

Acknowledgments

The thanks of the authors are particularly due to Mr. N. D. Macdonald and his staff on the sintering plants, who have done so much to provide sinter, of quality to which the furnaces did not object and in quantities to which the furnace operators could not object.

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The many furnace staff and men who had to learn so much in such a short time are also thanked.

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The Rating of Sinter Plants for Economic Output

By R. F. Jennings

SYNOPSIS

The major problems in designing sintering equipment to give maximum output at minimum costs ('economic rating') are discussed in relation to the conditions of system resistance obtainable with present methods of strand layering. A method is presented for calculating the basic design of a sintering strand to ensure the desired output of good blast-furnace sinter at minimum plant costs per ton. Examples are given of how the method applies to some typical practices. Although these examples do not exactly apply in specific instances, the recommendations should enable operators to ensure that their plant is designed and operated at the economic rating and maximum efficiency for their particular conditions. 836

THE value of a completely prepared and agglomerated burden for the blast-furnace is becoming more apparent in British practice. An agglomerated burden should lead to advantages in blast-furnace operation, by substantially removing unevenness in stack conditions.

A good sinter can be a perfect agglomerate if it is uniform in constitution, shows little carbon deposition, is porous and reasonably reducible, and does not decrepitate or break down in the upper part of the stack. There is particular scope for increasing the use of sinter prepared from ores containing a high proportion of moisture, carbon dioxide, and sulphur, and it may be economic to crush lump home ore for sintering, provided that the technical advantage of complete burden preparation can be matched by an equivalent reduction in overall costs per ton of pig iron.

The advantages obtainable from the use of good sinter are now being realized, at a time when the tonnages of available fines from both home and imported ore production are rapidly increasing. To deal with these increasing tonnages, a very great expansion of sintering capacity is at present envisaged or under construction in the United Kingdom.

These factors emphasize the importance of designing sintering equipment to give maximum output at minimum costs within the wide range of conditions found in U.K. sintering practice. The major problems arising in determining this 'economic rating' are discussed in this report

SINTERING PRACTICE IN THE UNITED KINGDOM

Sinter is produced in this country from a wide variety of iron-bearing materials. The physical and chemical differences between the resulting forms of sinter have been described by Wild[1] and Cohen.[2] The two extremes of practice are found in those plants operating wholly on home ores or on foreign concentrates. Other plants operate on various mixtures of home and foreign ores, including concentrates and pyrites. It is more common for such plants to operate with mixtures of many varying ingredients than on steady burdens.

Good-quality sinter can be made from almost all types of ore, but the process control is somewhat different in each case. For example, with a carbonate ore much work has to be done on the raw materials. This means that the process of sintering is generally slower, so that a higher proportion of the material should be passed a second time through the sintering machine as returned fines. A higher proportion of coke breeze also is usually necessary when sintering carbonate ore. On the other hand, it is usually more difficult to draw the sintering air through a bed of concentrates, and a higher suction is therefore required for a given rate of output. Moisture control is entirely different for concentrates as compared with home ores. Other variations between practices are discussed later in this paper.

In all practices, however, the rate at which the combustion air and flame front are drawn through the bed of raw material determines the rate at which sintering occurs.

THE SPECIFICATION OF A SINTERING MACHINE

Sintering is practicable as either a continuous or a batch process. For large tonnages, the continuous Dwight–Lloyd process is unquestionably more economical in cost and is technically more controllable. This report, therefore, concentrates on the design problems of the continuous machine, with only passing references to the batch process.

There are many important engineering problems to be considered in the specification of a sintering machine. One factor, however, is of overriding importance. The machine must be capable of producing the desired rate of output, and at minimum cost, from the materials to be treated.

The costs of sintering may be divided approximately as follows:

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Resource	Consumption Rate Per ton of blast-furnace sinter	Approximate Cost Per ton of blast-furnace sinter
Fuel:		
Coke breeze	14–31 therms	2*s.* to 5·5*s.*
Ignition gas	2 therms	0·85*s.*
Power:		
Fan suction and air volume	10–45 kWh.	0·85*s.* to 3·75*s.*
Labour:		
Operating	Manning 3 men per machine	About 1*s.*
Maintenance	?	Very variable
Capital		2*s.* to 5*s.*
	Total cost:	7*s.* to 16*s.*

Fuel is clearly a major item, but it is dependent primarily on the type of raw material to be sintered. The fuel cost per ton is independent of the rate of sintering or the output from a particular plant. Since this report is concerned with costs in relation to rating and output, it is possible to omit fuel from the sintering costs as they affect the selection and design of plant.

Labour is a smaller item in sintering costs, and since the manning of each machine is substantially constant the labour costs per ton for a given capacity or output can be considered as fixed and thus independent of sintering rate or size of machine. There may, of course, be circumstances in which the choice of plant is between, say, two large strands or three smaller strands. In these cases the labour charges will have to be included in the sintering plant costs. This particular aspect is discussed later.

Power costs and capital charges, on the other hand, are both major items directly dependent upon the rate of sinter output and the size of machine required for a given output. These factors are also interrelated, and this report suggests a method by which they may be correctly balanced in the plant specification to give a required output, from particular raw materials, at minimum costs per ton of sinter.

Now the rate of output from any sintering machine may basically be expressed in simple terms as follows:

Capacity of Sinter Machine

$$T \text{ (tons per unit time)} = \frac{V}{Q_s} \times A,$$

where V = rate of air flow per unit grate area of machine
Q_s = quantity of air required per ton of blast-furnace sinter
A = area of machine used for sintering.

This equation of capacity, in common with similar expressions for other processes, thus divides into three distinct components: V, concerned with rate of sintering; A, concerned with size of plant; and Q_s, concerned with raw materials. These components apply to the design of either a continuous strand or a batch pan type of machine. Each factor can be determined and measured independently. For a particular condition of raw materials, Q_s is substantially constant (this aspect is discussed in detail on p. 251). Therefore, the rate of output from given materials can be expressed as $T \propto V \times A$.

Clearly, the output of a plant can thus be determined within wide limits[3] by using sufficient suction to permit any desired rate of air flow to be drawn through the sinter bed, or by increasing the area of the strand. High suction, however, is expensive, as it requires increased power costs per unit of air volume. Similarly, a larger strand area, for a given output, will increase the capital charges per ton of sinter. Therefore, an optimum rate of air flow per sq. ft. of grate can be determined at which the combined capital charges and power costs per ton of blast-furnace sinter are at a minimum.

Thus, the economic capacity of a sinter plant can be expressed as:

Economic Capacity of Sinter Plant

= the rate of air flow per unit grate area multiplied by equivalent areas of machine at which the *combined* power costs and capital charges are at a minimum, divided by the air required per ton of sinter for the particular raw material to be sintered.

This relationship holds good provided that the sintering costs are considered in isolation. It may be that other factors, such as space limitations or rapid changes in output requirements to meet blast-furnace needs, influence the choice of rating. Such considerations, however, require individual assessments and cannot be considered inside the general problem of sinter plant specification.

Economic Rating of a Sinter Machine

It is thus necessary to determine the rate of air flow which will give minimum sintering costs with particular raw materials.

The rate at which air may be drawn through a sinter bed is governed by the permeability of the material, the bed thickness, and the applied suction. This relationship has been determined experimentally[4] as follows:

$$P \propto V\left[\frac{h}{s}\right]^n, \text{ or } V \propto P\left[\frac{s}{h}\right]^n,$$

where V = rate of air flow,* cu. ft./min. per sq. ft. of grate area
s = static suction, in. W.G.
h = bed height, in.
P = 'permeability,' B.P.U.
n = empirical index (usually about 0·6 for sinter beds).

Now, the theoretical power requirements of a given machine, and thus the annual power costs, are proportional to $s \times Q_s \times T$. Thus, for each ton of blast-furnace sinter produced from given materials the power costs are proportional only to suction. But $V \propto P\,[s/h]^{0\cdot6}$, and therefore:

$$V \propto (\text{power costs per ton})^{0\cdot6} \times \frac{P}{h^{0\cdot6}}.$$

The permeability and bed height used will vary between certain upper and lower limits according to sintering practice, and the ratio $h^{0\cdot6}/P$ may conveniently be called the 'system resistance' (S.R.).

System Resistance—The permeability of a sinter bed will vary according to the voidage obtainable in the material. This voidage will range between definite upper and lower limits according to the physical nature of the mixture, its moisture content, etc. The permeability is also determined by the effective particle size in the mixture; this also lies within definite limits. Thus, variations in permeability of the sinter

* Throughout this paper all volumes of air have been given at S.T.P. (30 in. of Hg, 60° F.).

Table I
SYSTEM RESISTANCE

Mix No.	Raw Materials (Principal Constituent)	Permeability, B.P.U.	System Resistance	
			Range	Average
1	Foreign concentrates	90	0·037–0·046	0·042
2	Foreign fine ores	105	0·036–0·043	0·039
3	Mixed home and foreign ores	120	0·034–0·039	0·037
4	Home ores (20% return fines)	110	0·037–0·043	0·040
5	Home ores (45% return fines)	130	0·032–0·037	0·035

bed are limited to a range of about 50%, according to the degree of mixing, the proportion of return fines, the nature of the sinter bond, and the moisture content of the raw material. In this paper the average permeability during the whole sintering process is considered. Recent experiments[4] have shown that the permeability changes during sintering and varies considerably along the strand, but since a continuous Dwight–Lloyd machine is operated at substantially constant suction, it is the mean permeability during sintering which determines the system resistance. For minimum system resistance the mean permeability should clearly be as large as possible.

Similarly, the bed should be as shallow as possible for minimum system resistance. However, there is a practical minimum to the height of bed which may be used, varying from 7 in. to 13 in, according to the type of materials to be sintered. In continental practice it is frequently considered that a shallow bed is detrimental to fuel economy. A deeper bed leads to more efficient heat exchange in its lower levels, so that a lower proportion of coke breeze or other fuel is needed in the lower half of the bed. In British practice, where coke breeze is plentiful and still fairly cheap, the bed height can be altered over a wide range to give acceptable system resistance consistent with the good sinter quality associated with deeper beds.

Recent production trials[5, 6] on plants in the U.K. have provided data for the measurement of typical system resistances obtained under widely different sintering conditions. These data suggest a range of values of system resistance (under the sinter strand) obtainable in good practice, and these values are shown in Table I with reference to specific practices.

The relationship between system resistance and permeability is shown in Fig. 1*a*. The dots represent typical system resistances calculated for different sintering practices, and the probable ranges of bed heights used in these practices are shown by the short arrowed lines. Thus, the average system resistances may be considered to vary from about 0·035 to 0·042, according to the type of mix to be sintered.

Using such values, the expression

$$V \propto \frac{(\text{power costs})^{0\cdot6}}{\text{system resistance}}$$

may be plotted. The effect of changes in the system resistance on the power costs is shown in Fig. 1*b*. As permeability increases and bed height is reduced, the power costs for a given rate of output will fall.

Such values of system resistance, however, will be obtained only under the best plant conditions. They imply correct control of moisture, bed height, and particle size. An even, regular rate of all feed materials from the bins through the mixer to the strand layering device (either swinging spout or roll feeder) is essential, and the layering must be free from compression or other hindrances to maximum initial permeability. Return fines must also be closely controlled in feed rate, size, and temperature, for maximum permeability during sintering.

When optimum conditions are not obtained the

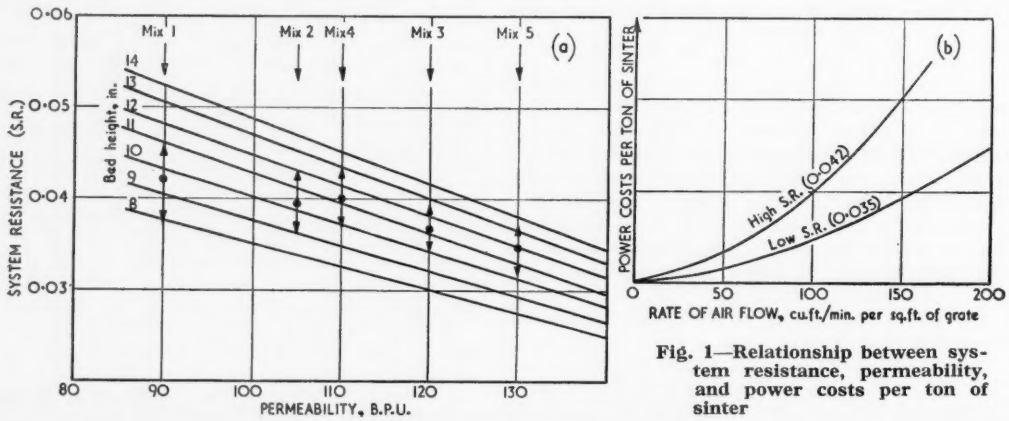

Fig. 1—Relationship between system resistance, permeability, and power costs per ton of sinter

system resistance will be greater, leading to decreased rate of output and increased costs per ton of sinter.

Air Required Per Ton of Sinter—The rate at which the flame front is drawn through the bed of raw material and the rate at which sintering is completed are both determined by the rate at which air is drawn through the sinter bed. Nevertheless, the quantity of air required to sinter a particular mix of raw materials cannot be defined by either chemical or physical calculations and appears to be unrelated to the combustion requirements of the fuel content of the mix.

It is necessary to measure this quantity for particular mixes. Recent observations, on a number of sinter plants,[5, 6] which may be considered to be typical of the range of U.K. practice are given in Table II. It is clear that there is a very wide range of air requirement per ton of blast-furnace sinter. This variation is much influenced by the loss of weight on sintering and by the proportion of return fines used in the sinter mixture. For example, it is possible (and common practice) to sinter home ores with 20% of return fines. Under these conditions, however, the sinter tends to be soft and the permeability of the sinter bed is low. For maximum output of good blast-furnace sinter it is better to use a highly permeable bed, obtained by including more return fines in the mix, even if this requires more air per ton.

The figures in Table II include a proportion of air loss from leakage under the strand seals when the sintering machine is operating at full rate and when sintering is just completed at the end of the strand. Further leakage may occur between the windlegs and the fan. Such losses would add to the total air requirement. It is possible to observe the same trend in air requirements for different mixes when sintering on a pan machine or on a small pilot plant. Corrections must, of course, be made for the difference in the proportion of leaks found on a moving Dwight–Lloyd strand compared with a well-sealed pan or sinter box.

All forms of leaks reduce sinter output and increase the cost per ton. This is particularly noticeable as a plant becomes older and clearances and seals tend to wear, unless this is remedied by constant maintenance.

SINTER COSTS

When considering the cost of making sinter on plants designed to produce between 250,000 and 750,000 tons of sinter per year, the two cost factors under the control of the designer are the capital charges (*e.g.*, size of plant per ton of sinter) and power costs (*e.g.*, suction and air rating per sq. ft. of grate). It is difficult to determine an exact value for the capital charges, since many unknown items are involved in a generalized case: *e.g.*, the expected return on capital, the expected life of the plant, and the amount of auxiliary equipment which may legitimately be included in the cost of a sintering machine. In any event, the capital charges per ton of sinter may be represented as a curve falling with increased rate of air flow. Power costs, on the other hand, rise with increasing suction and rate of air flow.*

As an example, the following cost assumptions have been made:

Capital charges at 15% per year on £200,000, plus £300 per sq. ft. of strand. (This is considered to include the cost of building, raw material bins, conveyors, mixers, moving-strand machine, dust collectors, fan, sinter screens, and cooler)
Power: 1*d.* per kWh. (1·34*d.* per h.p.hr.)

These assumptions may be applied to plant operating under the following conditions:

Plant output between 250,000 and 750,000 tons/yr.; plant operating 150 hr./week for 50 weeks/yr.
Air requirements per ton of blast-furnace sinter: 100,000 cu. ft.
System resistance between 0·035 and 0·042
Fan efficiency 70% (static), air volume at 150° C.

It is thus possible to combine the power costs per ton with curves of capital charge per ton based on rate of air flow per sq. ft. of grate area. The summation of power costs and capital charges, at each value of system resistance and output, gives the total plant cost per ton of blast-furnace sinter. This is

* Power costs do not include the pressure loss between strand and fan, which is considered to be a constant factor for all plants.

Table II

AIR REQUIREMENTS FOR MAKING SINTER FROM DIFFERENT RAW MATERIALS

Mix No.	Raw Materials (Principal Constituent)	Observed Air Required, cu. ft./ton of raw mix (Dwight–Lloyd Strand)	Loss of Weight on Sintering,* %	Return Fines in Mix, %	Air Required at End of Strand, cu. ft./ton of sinter	Air Required, cu. ft./ton of blast-furnace sinter
1	Foreign concentrates	80,000	11	20	90,000	116,000
2a	Soft hematite ores	45,000	15	20	53,000	70,000
			15	30†	53,000	96,000
2b	Hard magnetite ores	60,000	15	15	71,000	100,000
3	Mixed home and foreign ores	60,000	22	35†	75,000	133,000
4	Home ores	45,000	29	20	65,000	88,000
			26	35	62,000	115,000
			25	45†	60,000	150,000

* Including coke and sulphur burnt, moisture and volatiles evaporated
† Proportion of return fines for maximum permeability and best sinter quality

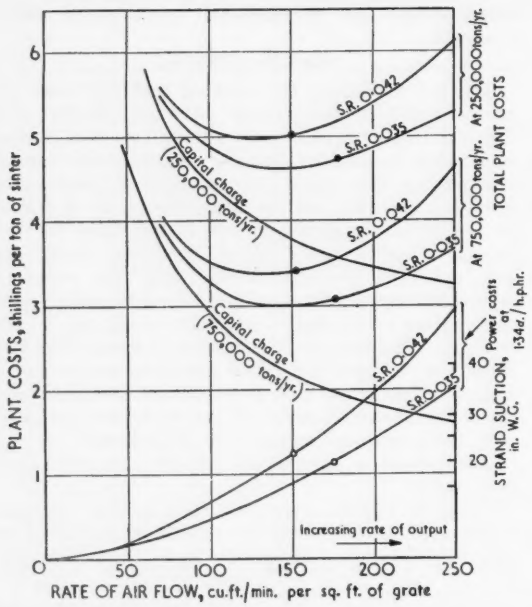
PLANT COSTS, shillings per ton of sinter
RATE OF AIR FLOW, cu.ft./min. per sq. ft. of grate
S.R. 0·042
S.R. 0·035
Capital charge (750,000 tons/yr.)
At 750,000 tons/yr.
At 250,000 tons/yr.
TOTAL PLANT COSTS
Power costs at 1·34d./h.p.hr.
STRAND SUCTION, in. W.G.
Increasing rate of output

Fig. 2—Costs per ton of sinter and plant rating at air requirements of 100,000 cu. ft./ton and 15% *p.a.* capital charges

shown in Fig. 2, which indicates how costs and air rating are affected by these variables. The following observations may be made:

(i) The curve for total plant cost per ton of sinter has a minimum value

(ii) The total plant cost per ton of sinter is fairly constant, and at a minimum value, over a reasonable range of ratings. The choice of economic air rating for a particular value of system resistance, etc., is thus arbitrary within certain limits. In this paper the 'economic air rating' is taken to be the rate of air flow corresponding to a rise in costs equal to 0·05*s.*/ton to the right-hand side of the minimum point

(iii) A higher system resistance always increases the cost per ton of sinter

(iv) A lower system resistance (achieved either by better permeability, lower bed height, or reduction of leaks) permits higher economic air rating (as defined above), as well as giving lower cost per ton

(v) Cost per ton rises steeply as plant size and output are reduced. (This is due to less-efficient use of total constructional costs)

(vi) The economic air rating of a plant is hardly affected by its size

(vii) Strand suction at the economic air rating changes with system resistance, but is not affected by plant output.

Now, both economic rating and costs per ton of sinter are related to the amount of air required per ton. Total costs and ratings rise with increasing air requirement, and further curves may be drawn showing characteristics similar to those in Fig. 2. The relationship between economic air rating and costs at various values of system resistance and air requirements is shown in Fig. 3.

The economic air rating will vary from 140 cu. ft./min. per sq. ft. at a system resistance of 0·042 and

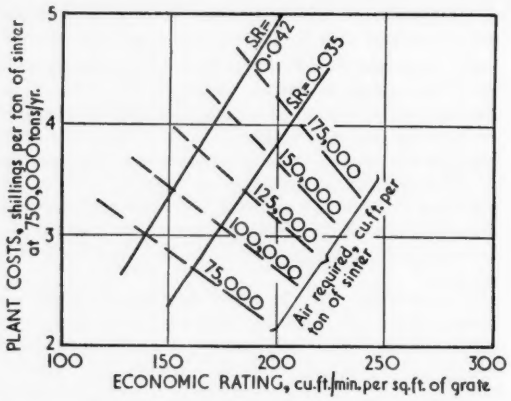

Fig. 3—Economic ratings and costs for various air requirements and values of system resistance

an air requirement of 75,000 cu. ft./ton, up to 210 cu. ft./min. per sq. ft. at a system resistance of 0·035 and an air requirement of 175,000 cu. ft./ton. The strand suctions at the economic rating will vary from 19 in. W.G. to 28 in. W.G., according to system resistance. These figures emphasize that the economic air rating for sintering all types of material is not appreciably affected by the output of the machine, but is principally dependent on two factors:

(i) The system resistance of the sinter bed
(ii) The air requirement per ton of sinter.

On the other hand, total plant costs per ton of sinter at the economic air rating are severely affected by three factors:

(i) The yearly output. Costs per ton increase steeply with decreasing plant output, approximately according to the relation

$$\text{Cost } (s./\text{ton}) = \text{Cost at } 750{,}000 \text{ tons/yr.} + 0{\cdot}8\left(\frac{750{,}000 - \text{yearly tonnage}}{\text{yearly tonnage}}\right)$$

(ii) Air required per ton. Costs fall by about one-quarter if the air required falls from 150,000 to 75,000 cu. ft./ton

(iii) System resistance. Costs fall by one-tenth as the system resistance decreases from 0·042 to 0·035.

These ratings and costs will, of course, change if returns on capital or power costs per kWh. are changed. A lower return on capital will reduce cost per ton and lower the economic air rating. A reduced cost of power will reduce cost per ton but raise the economic air rating.

CALCULATION OF ECONOMIC RATING AND STRAND SUCTION FOR PARTICULAR RAW MATERIALS AND OUTPUT

For a new machine designed to make a known yearly tonnage it is possible to calculate the economic air rating and the strand suction required at this rating, if the expected system resistance and the air required per ton of sinter can be estimated for given materials. It is also possible to estimate the probable plant costs per ton of sinter.

Table III gives some typical calculated ratings, suctions, and costs. These calculations give *minimum* costs and strand areas for outputs of 250,000 and 750,000 tons/yr., based on the use of 90% of strand

Table III

ECONOMIC AIR RATINGS, SUCTIONS, PLANT COSTS, AND STRAND AREAS FOR CONTINUOUS SINTERING MACHINES

Mix No.	Ore Type (Principal Constituent)	Return Fines, %	Optimum System Resistance Under Strand	Air Required, cu. ft./ton of sinter		Economic Rating, cu. ft./min. per sq. ft. of grate	Strand Suction at Economic Rating, in. W.G.	Power Required, h.p.hr./ton of sinter†	Total Plant Cost at Economic Rating, shillings/ton of sinter		Output Rating, tons of sinter per day per sq. ft. of grate	Maximum Output from 1000-sq. ft. Machine at Economic Air Rating, tons/yr.
				Estimated	+ 10%				At 250,000 tons/yr.	At 750,000 tons/yr.		
1	Foreign concentrates	20	0·042	115,000	125,000	165	25	18·2	5·3	3·7	1·92	590,000
2	Soft hematite ore	20	0·039	70,000	77,000	150	18·5	8·3	4·4	2·8	2·80	870,000
3	" " "	30	0·037	96,000	105,000	170	21·5	13·2	4·8	3·2	2·35	730,000
4	Hard magnetic ore	15	0·041	100,000	110,000	160	23	14·8	5·1	3·5	2·10	650,000
5	Mixed home and foreign ore	35	0·038	133,000	150,000	185	26	22·6	5·5	3·9	1·80	550,000
6	Home ore	20*	0·040	88,000	100,000	160	22	12·8	4·9	3·3	2·30	720,000
7	" "	35	0·038	185,000	125,000	175	23·5	17·0	5·2	3·6	2·00	630,000
8	" "	45	0·035	150,000	165,000	205	27	26·0	5·5	3·9	1·80	560,000

* Generally poor sintering conditions unless substantial quantities of other materials, such as mill scale, can be added to the mix
† Calculated for suction under strand: air at 150° C., fan efficiency 70%

air for sintering. It is usual to allow some grate area as a margin (to prevent discharge of unsintered material), and it may be assumed that at least 10% of total fan air passes this part of the strand. Thus, total air requirements for sintering may be assumed to rise 10%.

The reduced system resistance obtained by using more return fines helps to offset the higher air requirement of mixes containing a high proportion of return fines, but it cannot reduce power costs sufficiently to prevent some rise in total costs and decrease in capacity of the machine. It is considered that home ores cannot generally be sintered satisfactorily with less than 30% of return fines unless other materials are added to give improved strength to the sinter, because the use of more return fines with materials forming a soft sinter helps to improve the strength of the product. Calcination and sulphur removal are also more complete.

Engineering difficulties and space limitations will probably limit the maximum strand area to 1000 sq. ft. on a machine 6 ft. wide. If an output of 750,000 tons/yr. is required, some ores cannot be sintered at economic ratings to produce this tonnage on a single strand of this width. Under these conditions, the increased costs at higher ratings necessary to achieve 750,000 tons/yr. on a single 1000-sq. ft. strand may be compared with the increase in cost per ton if two smaller strands were operated at economic air ratings and lower outputs.

For example, suppose 750,000 tons of sinter per year are required from mix No. 5. This can be produced from two machines, each of about 670 sq. ft. grate area and capable of making 375,000 tons/yr. at an economic rating of 185 cu. ft./min. per sq. ft. and a total plant cost of 4·7*s.*/ton. But a single 1000-sq. ft. machine can produce 550,000 tons/yr. from this mix at an economic rating of 185 cu. ft./min. per sq. ft. and a plant cost of 4·2*s.*/ton. To produce 750,000 tons/yr. from this one machine would require a rating of $185 \times 75/55 \simeq 250$ cu. ft./min. per sq. ft. (strand suction = 43 in. W.G.). At this rating and output the total plant costs would be about 4·6*s.*/ton.

Similarly, for mix No. 1 (concentrates) two machines each of 640 sq. ft. grate area and capable of making 375,000 tons/yr. would operate at an economic rating of 165 cu. ft./min. per sq. ft. and a plant cost of 4·5*s.*/ton. A 1000-sq. ft strand can produce 590,000 tons from concentrates at an economic rating of 165 cu. ft./min. per sq. ft. and a cost of 3·9*s.*/ton. To produce 750,000 tons/yr. from this machine would require a rating of $165 \times 75/59 = 210$ cu. ft./min. per sq. ft. (strand suction = 37 in. W.G.), and at this rating and output the plant costs would be about 4·1*s.*/ton.

Under these conditions it would be more economical to over-rate one 1000-sq. ft. machine to give extra output above economic rating than to use two smaller machines for the same output at economic air rating. Moreover, there would be a small saving in labour costs per ton. Nevertheless, this method will be more costly per ton than when operating the 1000-sq. ft. machine at economic rating (*see* Fig. 4). For very large tonnages there is thus considerable incentive towards the development and use of wider

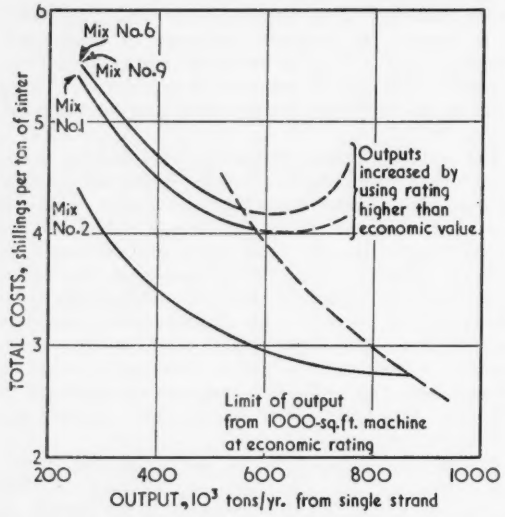

Fig. 4—Relationship between costs per ton of sinter and machine output on a grate area of 1000 sq. ft.

strands, up to 12 ft. in grate width, provided that the engineering problems of pallet warping and expansion can be overcome.

For the design of new continuous sintering machines it is thus possible to draw the following conclusions in regard to economic rating and costs:

(*a*) When the strand would have to be not greater than 1000 sq. ft. in area for the output required.

(i) The air rating corresponding to lowest total plant costs (*i.e.*, the economic air rating) will vary from 150 cu. ft./min. per sq. ft. for soft hematite ores up to 205 cu. ft./min. per sq. ft. for home ores. The rating increases with increasing proportion of return fines in the raw mix. These ratings are substantially independent of the designed output tonnage from the machine

(ii) The strand output at economic rating will vary between 1·80 and 2·80 tons/day per sq. ft. of grate, according to air requirements of different materials

(iii) Strand suction should, at these ratings, vary from 18·5 to 27 in. W.G., according to the type of ore to be sintered

(iv) The plant cost at economic air rating decreases sharply with increased size of machine, varying from 4·4 to 5·5 *s.*/ton at 250,000 tons/yr. and from 2·8 to 3·9 *s.*/ton at 750,000 tons/yr.; the difference in costs at each rate of output depends on the type of raw materials. An increase in the proportion of return fines substantially increases the cost per ton.

(*b*) When the output required is greater than can be produced from a grate area of 1000 sq. ft. at economic air rating.

The operating rating may be increased to 250 cu. ft./min. per sq. ft. and strand suction may be up to 43 in. W.G. This would give the required output at a lower cost per ton than that obtainable from two smaller strands operating at economic rating.

In all cases the suctions and ratings must match the expected system resistance of the sinter bed.

These ratings and costs apply only if the suggested system of capital charges and power costs is accepted.

COMPARISON OF ECONOMIC AIR RATINGS WITH ACTUAL DESIGN RATINGS OF U.K. PLANTS

Data for 14 Dwight–Lloyd strands built or being built in this country are compared in Table IV. Machines *B*, *G*, and *M* are rated according to the suggested economic rating; *i.e.*, the ratio design-rating/economic-rating is approximately unity. Nine plants (*A*, *C*, *D*, *F*, *I*, *J*, *K*, *L*, and *N*) are more conservatively rated, and the remaining two (*E* and *H*) are rated too high for maximum economy on this basis. Only three of the machines (*G*, *I*, and *K*) have fan design points closely matched to the system resistance that is expected with the materials to be sintered. In eight cases (*A*, *B*, *C*, *D*, *E*, *F*, *L*, and *N*) the machines have been designed for too high a system resistance, and in three cases (*H*, *J*, and *M*) too low a system resistance has been allowed. The probable system resistance is matched to an economic air rating in only one machine (*G*).

These variations in data suggest that sinter-plant design is not always based on the form of analysis suggested in this report. Incorrect air ratings and wrongly estimated system resistances may result in output below that required and costs per ton above the minimum.

If the design system resistance is higher than that obtained in practice (*i.e.*, S.R. ratio greater than unity), the operating air rating of the machine will be increased above the design point. This will give greater output than was expected, provided that the fan motor can deal with any extra power required. Under these conditions plants *A*, *C*, *D*, *F*, *L*, and *N* could obtain greater output than as designed, and they could operate closer to their economic air ratings and thus at lower plant cost per ton. Maximum economy at the new operating points, however, can only be obtained by fitting a different fan, designed to match the actual system resistance and rating, at high efficiency.

If the design system resistance is lower than that obtained in practice (*i.e.*, S.R. ratio less than unity), the operating air rating of the machine will be decreased below the design point. Output will thus be below that expected. The plant costs per ton at lower output may be less if the air rating ratio is greater than unity. Under these conditions the outputs for plants *H*, *J*, and *M* would be less than as designed, and the costs per ton might fall slightly at plants *H* and *M* and rise at plant *J*. Outputs can be raised to the original designed values by fitting a different fan with greater suction, but this will increase both power costs and capital costs, thus substantially increasing plant costs per ton. Where dustcatchers are fitted it may be possible to modify their design to transfer part of the pressure originally used in the cyclone to increase the suction under the strand and so increase the design rating to economic level. This has been done successfully in at least one plant.

ADJUSTMENT OF OLDER MACHINES TO CHANGED SINTERING CONDITIONS

In U.K. sintering practice the materials treated in a particular plant often change from time to time in accordance with available supplies and the relative costs of different iron-bearing raw materials. Also,

older machines were sometimes rated lower than more modern plants, so that old machines may now be called on to produce more sinter than the output specified in the original design. To meet either of these requirements satisfactorily it is necessary to re-rate the potential performance of the machine by the use of a new fan, designed to meet the changed conditions at maximum efficiency.

If a machine is old enough for capital charges to be fully written off, it may be greatly advantageous to increase the suction and rating well above the economic point. This will give increased output at lower total plant cost than if a further machine were built to deal with the increased tonnage. However, it is not possible to say how far this idea would be acceptable to managements, as the apparent costs of sintering as shown by power costs would substantially rise.

Moreover, if leaks between the sinter strand and the fan are allowed to develop with age of the machine, they may give a deceptive picture of the actual system resistance of the machine, and might even lower the rate of sinter output and substantially increase the power costs per ton. Care must therefore be taken in assessing the actual system resistance and correct rating for an old machine.

MATCHING FAN CHARACTERISTICS TO SINTER STRAND CONDITIONS

The foregoing discussion emphasizes that the performance of the fan is of great importance to any sinter plant. Unless the fan characteristics can

Table IV

ACTUAL DESIGN RATINGS AND SUGGESTED ECONOMIC RATINGS FOR U.K. CONTINUOUS SINTERING PLANTS

Plant Code Letter	Plant Data at Fan Design Point				Calculated System Resistance $(4)^{0\cdot4}\div(3)$	Estimated Raw Material Conditions			Optimum Operating Conditions		Rating Ratio (3)÷(11)	S.R. Ratio (6)÷(10)	Remarks
	Grate Area, sq. ft.	Air Rating, cu. ft./min. per sq. ft.	Suction, in. W.G.			Raw Material Type	Return Fines (Current Practice), %	Air Required, cu. ft./ton of blast-furnace sinter	Probable System Resistance (see Table III)	Economic Air Rating, cu. ft./min. per sq. ft.			
			Strand	Fan									
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)	(13)	(14)
A	306	133	21	24	0·047	Foreign ore	15	110,000	0·041	160	0·83	1·15	Dustcatcher with 1-in. W.G. pressure drop
B	970	172	40	45	0·053	Foreign concentrates	20	125,000	0·042	165	1·04	1·26	Dustcatcher with 3-in. W.G. pressure drop. Variable-speed fan fitted
C	440	112	18	23	0·050	Foreign ore	20	110,000	0·041	160	0·70	1·22	Dustcatcher with 3-in. W.G. pressure drop
D	648	135	25	27	0·056	Mixed home and foreign ore	35	150,000	0·040	185	0·68	1·40	No cyclone dustcatcher
E	450	175	33	35	0·046	Home ore	20	100,000	0·040	160	1·10	1·15	No cyclone dust catcher
F	450	115	21	23	0·054	,, ,,	,,	,,	,,	,,	0·72	1·35	,, ,, ,,
G	615	158	22	24	0·040	,, ,,	,,	,,	,,	,,	1·00	0·99	,, ,, ,,
H	762	182	20	27	0·033	,, ,,	,,	,,	,,	,,	1·14	0·82	Dustcatcher with 5-in. W.G. pressure drop. Variable-speed fan fitted
I	648	150	16	21	0·035	Home ore	45	165,000	0·035	205	0·74	1·00	Dustcatcher with 3–4-in. W.G. pressure drop
J	720	168	16	21	0·031	,, ,,	,,	,,	,,	,,	0·82	0·89	,, ,, ,,
K	1000	183	24	26	0·036	,, ,,	,,	,,	,,	,,	0·90	1·03	No "cyclone" dustcatcher
L	600	107	18	23	0·052	Home ore	20	100,000	0·040	160	0·67	1·30	Dustcatcher with 3-in. W.G. pressure drop
M	720	168	22	27	0·038	Foreign ore	20	110,000	0·041	160	1·05	0·93	Dustcatcher with 4-in. W.G pressure drop
N	567	126	22	27	0·049	Foreign ore	20	110,000	0·041	160	0·79	1·19	Dustcatcher with 4-in. W.G. pressure drop

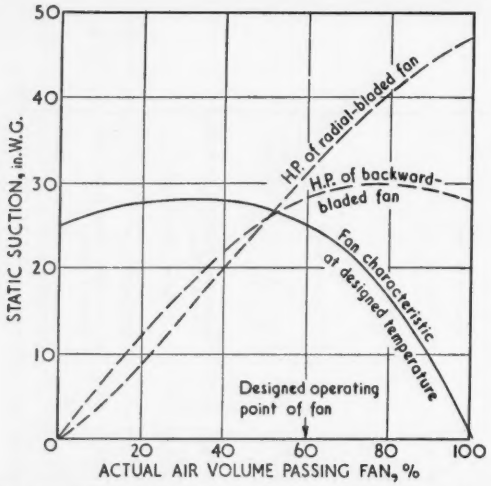
STATIC SUCTION, in.W.G.
ACTUAL AIR VOLUME PASSING FAN, %
H.P. of radial-bladed fan
H.P. of backward-bladed fan
Fan characteristic at designed temperature
Designed operating point of fan

Fig. 5—General characteristics of sinter-plant fans

match the rating and system resistance necessary for a desired output, the performance of the whole plant is bound to suffer.

Fans for sinter plants must be robust, simple, and capable of consistent operation for many months without major repair or replacement of the blading, while handling a flue gas which may contain several grains of dust per cu. ft. at suctions up to 40 in. W.G. and temperatures up to 200° C. Such conditions restrict sinter plant fans to the single-stage centrifugal type. Fan blading may be purely radial in form, or it may be backward-curved blading. The latter is likely to have a somewhat shorter working life, as tip speed is greater, but the efficiency of this kind of blading is sufficiently high, over a wide range of operation, for such fans to be self-limiting in power requirements. Typical characteristics for radial and backward-curved fans are shown in Fig. 5.

The self-limiting power characteristics coupled with the somewhat higher efficiency obtained with backward-curved fans is of great value in sinter plant operation. With such a fan the plant will not be limited in output by lack of motor power, provided that the fan temperature can be maintained near its design value. Full advantage can thus be taken of improvements in system resistances obtained by efficient process control or changes in raw material conditions.

However, if the fan temperature substantially falls, say from the design value of 150° to 100° C., the fan motor may not be capable of accepting the increased load so imposed. This increase in load occurs because the actual volume passing through a fan is unchanged by temperature, while with falling temperature the suction values of the fan characteristic will rise (*see* Fig. 6). Thus, for a given system resistance, the fan can only match the system-resistance curve at a higher suction and higher S.T.P. gas flow. This needs more power, which may not be available in the fan motor, and vane control must be used to maintain the mass air flow within the power of the fan. On the other hand, of course, if ample fan power is available this will give an increased rate of production without vane control.

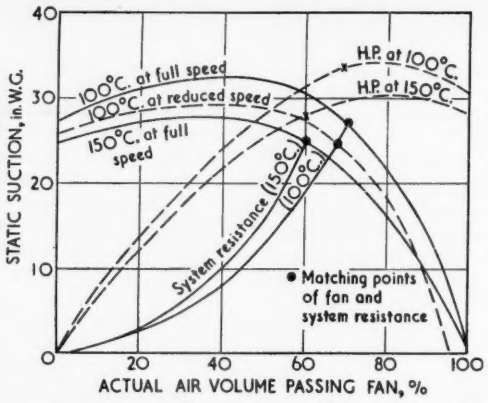

Fig. 6—Effect of temperature and fan speed on matching of system resistance and fan characteristic

If the speed can be altered, by using a variable-speed coupling between the motor and the fan shaft, the fan can be matched at the lower temperature to maintain an optimum rate of S.T.P. air flow at high efficiency, and so to produce sinter at maximum rate consistent with lowest overall costs.

It is therefore strongly recommended that variable-speed fans of self-limiting power characteristics should be fitted to future plants. This applies particularly to those plants making sinter from variable raw materials and mixed burdens. It is considered that their advantages greatly outweigh the disadvantages arising from engineering complications and the need for good dustcatcher design.

PROBABLE FUTURE CHANGES IN SINTER PLANT RATING

So far, the economic rating of sinter plants using a variety of raw materials has been considered in relation to the conditions of system resistance obtainable with present methods of strand layering.

There are very substantial advantages to be obtained from reducing sintering costs and increasing rates of output by improving sinter bed permeability beyond that already obtainable. New methods of strand layering, devised to give initially a more permeable sinter bed, combined with drying out the layered raw mix before ignition so as to increase the average permeability during sintering, have yet to be applied to plant design.

It is expected that the use of such ideas with raw materials that readily pelletize will enable the economic rating of plants to be substantially raised in future designs, with corresponding reductions in plant costs per ton of sinter.

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Effect of Chromium on the Thermodynamic Activity of Carbon in Liquid Iron

By F. D. Richardson and W. E. Dennis

SYNOPSIS

A study has been made of the influence of chromium on the thermodynamic behaviour of carbon in molten iron. The equilibrium

$$[C]_{\text{in alloy}} + CO_2 = 2CO$$

has been investigated in the temperature range 1560–1660° C. at chromium concentrations of 10–80%. The results are expressed in terms of the activity coefficient of carbon, which is markedly lowered by the chromium; e.g., at 4 at.-% of carbon $\gamma_{[C]}$ at 1500° C. is decreased by the factor 10·7 on the addition of 45% of chromium. Approximate values have been derived for the partial molar heats of solution of carbon in the alloys: these change from +6.4 kg. cal. per gramme-atom in pure iron to − 5.3 kg. cal. in an alloy containing 45% of chromium. The results are interpreted in terms of a clustering together of chromium and carbon atoms in the molten iron. 800

THE pronounced affinity of chromium for carbon suggests that the activity coefficient of carbon dissolved in liquid iron should be significantly lowered by the addition of chromium. A proper understanding of refining processes involving iron–carbon–chromium melts thus requires a quantitative knowledge of this effect, which has apparently not been previously studied.

Before the ternary alloys were investigated, the simple iron–carbon system was re-examined, as previous work did not provide data of sufficient scope or accuracy to allow satisfactory thermodynamic calculations to be made. This preliminary work has already been reported in detail,[8] so that only the activities obtained from its results will be considered in the present paper.

EXPERIMENTAL DETAILS

Outline of Method Chosen

To determine the activity of carbon in liquid iron–chromium–carbon alloys, it was decided to study the equilibrium with a gaseous phase in which the carbon activity could be varied over a suitable range. Mixtures of CO and CO_2 were selected for this purpose because the equilibria between these gases and pure β-graphite have been accurately determined,[10] and because the CO/CO_2 ratios required for this investigation were not too large. Thus, the carbon activities were obtained from measurements of the equilibrium

$$[C] + CO_{2(gas)} = 2CO_{(gas)} \quad \ldots\ldots\ldots\ldots\ldots\ldots (1)$$

where [C] represents carbon dissolved in the molten alloys. This was investigated at 1560°, 1660°, and 1760° C. for ratios of CO/CO_2 up to 800/1.

In each experiment a bead of liquid alloy was brought into equilibrium with a gas stream of known composition, quenched, and analysed for carbon. Methods in which the flowing gas is brought to equilibrium with an alloy of known composition are not applicable to this system. This is because the gases, after passing the alloy, deposit carbon at a significant rate during cooling, owing to the catalytic effect of the iron which condenses at the exit end of the reaction tube. In almost every instance equilibrium was approached from both sides, *i.e.*, from initial carbon contents above and below the equilibrium value.

In most details, the experimental technique was the same as that already described by the authors for their work on carbon in pure iron.[8] The equilibrium apparatus is shown in Fig. 1. The methods of making and analysing the beads are described below, as they were somewhat different from those used for the other work.

Alloys

In preliminary experiments it was found that the rate of approach to equilibrium was slow and largely controlled by the capacity of the gas to transport carbon to or from the liquid metal. To reach equilibrium from both sides in reasonable periods of time, each 2-g. charge was made up of pieces taken from high- and low-carbon ingots, so as to give an initial composition about 0·1% C above or below the expected equilibrium value.

All the ingots (100-g.) used during the investigation

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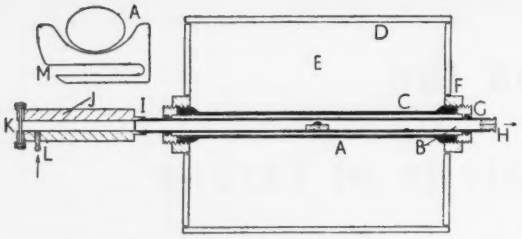

A. Refractory boat containing liquid bead
B. Recrystallized alumina reaction tube
C. Graphite resistor tube
D. Sindanyo furnace case
E. Thermal insulation (fine carbon)
F. Water-cooled brass electrodes
G. Threaded contact blocks with steam packing glands
H. Rubber bung with radiation shield and gas exit tube
I. Water-cooled brass fillet sealed with picene wax to reaction tube
J. Water-cooled quenching tube
K. Heat-resistant optical window
L. Gas inlet
M. Radiation hole for sighting optical pyrometer

Fig. 1—Equilibrium apparatus

were made by melting high-purity iron, graphite, and chromium in magnesia crucibles, under a vacuum or 99·8% argon in a high-frequency induction furnace. The method used was to prepare high-carbon (about 5% C) ingots by melting the iron in graphite crucibles and, after analysis, to use pieces of this 'carburized iron' to make up the required charges with iron and chromium. This duplex melting procedure was necessary because direct melting of the 'pure' iron and chromium, both of which have high oxygen contents, with pieces of graphite gave a 'wild heat' in which the composition changed in an unpredictable manner owing to spitting and the ejection of small fragments of undissolved carbon. After about ½ hr., with a superheat of at least 100° C., the alloy melts were rapidly cooled, and in this way the segregation of both carbon and chromium was kept within reasonable limits.

The analyses of the original iron* and of the chromium† are given in Table I, together with some analyses‡ of the alloys after equilibration. After being melted to make the ingots and used in the equilibrium experiments, the material had picked up about 0·006% of silicon and 0·05% of sulphur. When alumina boats were used, about 0·15% of aluminium was found in the alloy (as a result of reduction of Al_2O_3), as was expected from the work of Geller and Dicke.[3]

There were no other significant impurities.

Analysis

For the carbon determinations the B.S. 1121 combustion method was used; in this, oxygen is passed over a boat containing the sample and a suitable flux at 1200 ± 10° C., and the CO_2 formed is absorbed and weighed in a Nesbitt bulb containing Sofnolite and Anhydrone. Samples containing chromium were crushed in a tungsten-carbide mortar and intimately mixed with 'carbon-free' iron of the same mesh inside a sheath of lead foil, to ensure complete oxidation. Samples prepared by crushing gave the same results as those prepared by filing, where filing was possible. This method gave a reproducibility of ± 0·01% C, except with samples containing more than 45% of Cr, for which it was ± 0·02%. The chromium analyses were done‡ by the B.S. 1121 (Part 18, 1951) method, which gave a reproducibility of ± ½% of the chromium percentage in the alloy. Chromium analyses by the same method carried out by the authors on some preliminary ingots were in good agreement with their results.

Chromium Loss during Equilibration

The vapour pressure of chromium is much greater than that of iron at the temperatures encountered in this work. Thus, it might be expected that during equilibration the proportion of chromium to iron in the alloy would decrease. A faint purple discoloration of the refractory boats (increasing in intensity with the chromium content in the bead) indicated a further means by which chromium could be lost.

During preliminary experiments, it was found that in beads initially containing 40% of Cr a decrease of only 1% of Cr occurred during 1·3 hr. at the highest temperature (1760° C.). In view of the ± 3% probable error in the results obtained with the simple binary system, it was decided that the effect of chromium losses occurring during equilibration could be neglected. However, when the experimental work on the ternary system had been completed, it seemed desirable that this factor should be taken into account for two reasons: experiments lasting for quite long periods (up to 5 hr.) had been done and the overall accuracy of the results on the ternary system appeared to be better than expected.

To enable the chromium losses that had occurred during equilibration to be estimated, 16 experiments were repeated and the specimens analysed for chromium. From these results the effect of time, composition, and temperature upon this loss (found to be

Table I

ANALYSES OF MATERIALS BEFORE AND AFTER INVESTIGATION

Impurities, wt.-%	Iron Used for Ingot Manufacture	Chromium Used for Ingot Manufacture†	Material after Equilibration
Cr	0·001	s.p.	<0·01
C	0·005	s.p.	0–3
O	0·2	0·2	<0·01
Si	0·01	s.p.	<0·01
Al	0·001	s.p.	0·015
Ni	0·010	s.p.	<0·01
P	0·001	s.p.	<0·01
S	0·010	s.p.	<0·05
Mn	0·005	s.p.	<0·01
Cu	0·006	0·0005	<0·001
Mg	*	trace	<0·001
Co		s.p.	<0·02
Ti		s.p.	<0·02
V		s.p.	<0·01
W		s.p.	<0·02
Mo		s.p.	<0·01
Zr		s.p.	<0·001
Sn		s.p.	<0·001
Pb		s.p.	<0·001

* No analysis available
† s.p. indicates that no trace could be detected spectrographically

* Supplied by the British Iron and Steel Research Association.
† Supplied by Johnson, Matthey and Co., Ltd.
‡ Carried out by the B.I.S.R.A. section of the Bragg Laboratory.

Table II*a*

EXPERIMENTAL RESULTS

Charges made up to give approx. 10% of Cr
Maximum uncertainty in $Cr_{initial} = \pm 0{\cdot}07\%$
" " " $Cr_{final} = \pm 0{\cdot}27\%$

Temp., °C.	$p(CO)^2/pCO_2$	Time, hr.	Final Cr, %	Initial C, %	Final C, %
1560	201	2·0	9·61	0·68	0·54
"	201	2·0	9·65	0·38	0·51
"	403	2·0	9·59	0·85	0·90
"	403	2·0	9·57	0·95	0·95
"	604	2·0	9·55	1·15	1·20
"	604	2·0	9·53	1·24	1·18
"	805	3·0	9·51	1·47	1·44
1660	198	1·0	9·63	0·38	0·355
"	300	2·5	9·37	0·38	0·48
"	300	2·0	9·43	0·51	0·515
"	398	1·5	9·50	0·76	0·63
"	398	1·5	9·53	0·56	0·66
"	696	1·5	9·48	0·90	0·92
"	800	3·0	9·21	0·90	1·09
"	800	3·0	9·18	1·10	1·09
1760	300	2·0	9·10	0·38	0·35
"	450	2·0	9·10	0·38	0·515
"	603	2·0	9·06	0·65	0·63
"	603	2·0	9·10	0·38	0·65
"	802	2·0	9·03	0·92	0·775

Table II*c*

EXPERIMENTAL RESULTS (contd.)

Charges made up to give approx. 50% of Cr
Maximum uncertainty in $Cr_{initial} = \pm 0{\cdot}64\%$
" " " $Cr_{final} = \pm 1{\cdot}64\%$

Temp., °C.	$(pCO)^2/pCO_2$	Time, hr.	Final Cr, %	Initial C, %	Final C, %
1560	200	5·0	45·24	2·50	2·30
"	200	5·0	44·85	1·40	1·48
"	201	5·0	45·07	2·00	2·19
"	251	5·0	45·24	2·50	2·58
"	252	5·0	45·45	3·05	2·69
"	350	5·0	45·24	2·49	2·85
"	352	5·0	45·45	3·05	3·15
1660	198	2·0	47·84	1·20	1·35
"	201	2·0	47·59	1·55	1·37
"	302	2·0	47·68	1·80	1·89
"	403	3·0	47·25	2·41	2·42
"	501	3·3	47·22	2·91	2·74
"	501	3·0	47·33	2·61	2·70
"	601	3·0	47·49	3·05	3·11
1760	200	4·0	46·12	1·08	0·77
"	201	3·0	46·61	0·65	0·89
"	301	3·0	46·78	1·09	1·18
"	302	3·0	46·89	1·41	1·27
"	450	3·0	47·01	1·74	1·62
"	448	3·0	47·01	1·75	1·67
"	599	3·0	47·13	2·06	2·18
"	599	3·0	47·23	2·35	2·31
"	796	3·0	47·32	2·60	2·72

practically independent of $(pCO)^2/pCO_2$) were evaluated. These losses were never greater than 10% of the final chromium content and could be estimated for any set of conditions to within $\pm$ 20%, *i.e.*, the final error involved in estimating these corrections was never greater than $\pm$ 2%.

Table II*b*

EXPERIMENTAL RESULTS (contd.)

Charges made up to approx. 25% of Cr
Maximum uncertainty in $Cr_{initial} = \pm 0{\cdot}12\%$
" " " $Cr_{final} = \pm 0{\cdot}62\%$

Temp., °C.	$p(CO)^2/pCO_2$	Time, hr.	Final Cr, %	Initial C, %	Final C, %
1560	202	2·0	24·56	0·98	0·96
"	202	2·0	24·60	0·88	0·97
"	301	3·0	24·36	1·55	1·42
"	301	2·0	24·44	1·31	1·29
"	405	2·0	24·41	1·40	1·60
"	402	2·0	24·27	1·80	1·73
"	402	2·0	24·35	1·59	1·68
"	504	2·0	24·20	1·99	1·87
"	504	2·0	24·28	1·85	1·90
"	600	3·0	24·09	1·94	2·05
1660	200	2·0	24·35	0·50	0·65
"	200	2·0	24·25	0·76	0·73
"	400	3·0	23·77	1·27	1·17
"	400	3·0	23·87	0·99	1·13
"	796	3·0	23·54	1·90	1·96
1760	198	2·0	23·78	0·18	0·34
"	198	2·0	23·63	0·60	0·52
"	200	2·0	23·68	0·45	0·45
"	399	2·0	23·58	0·74	0·855
"	603	3·0	22·87	1·05	1·105
"	603	3·0	22·80	1·25	1·18
"	804	3·0	22·76	1·35	1·42

Table II*d*

EXPERIMENTAL RESULTS (contd.)

Charges made up to give approx. 75% of Cr
Maximum uncertainty in $Cr_{initial} = \pm 0{\cdot}39\%$
" " " $Cr_{final} = \pm 2{\cdot}0\%$

Temp., °C.	$(pCO)^2/pCO_2$	Time, hr.	Final Cr, %	Initial C, %	Final C, %
1760	201	4·0	74·35	1·88	2·32
"	201	3·0	76·04	2·90	2·68
"	202	3·0	75·77	2·50	2·48

RESULTS

The equilibrium conditions for the reaction expressed in equation (1) were determined at 1500°, 1600°, and 1760° C. for values of $(pCO)^2/pCO_2$ up to 800/1, with alloys containing up to 50% Cr. Three experiments with an 80% Cr alloy were done at 1760° C. for a $(pCO)^2/pCO_2$ value of 200/1, to ascertain whether data for the highest temperature could be extrapolated for higher chromium contents.

Seventy-six separate equilibrium studies were made with the ternary system, excluding preliminary tests, experiments to measure the loss of chromium during equilibrations, and measurements to determine the effect of dissolved nitrogen (*see* next section).

The results of the equilibrium experiments are given in Table II*a–d* and are represented in Figs. 2 and 3. The relation between the equilibrium carbon content and the chromium content at a given temperature is represented by a family of constant $(pCO)^2/pCO_2$ curves. In these figures the change in the carbon content during an experiment is represented by an

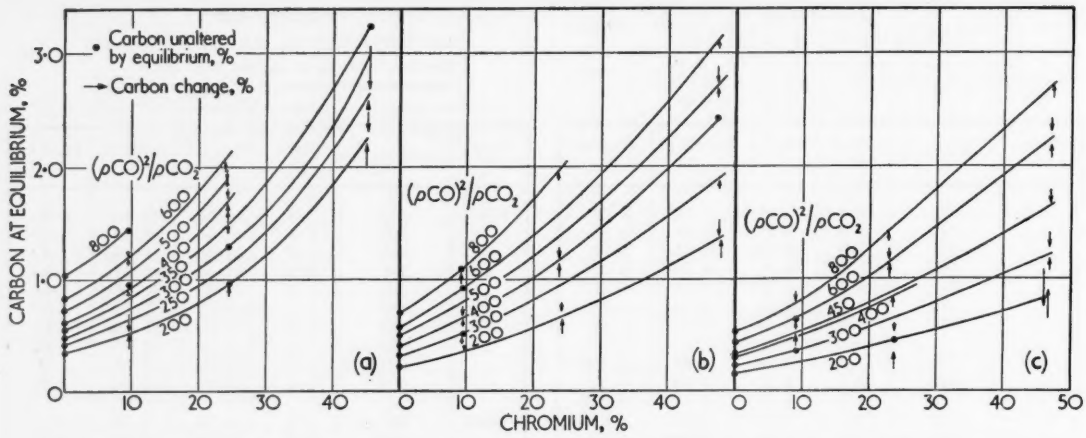

Fig. 2—Percentages of carbon in equilibrium with various $(pCO)^2/pCO_2$ ratios at (*a*) 1560°, (*b*) 1660°, and (*c*) 1760° C., as a function of chromium content

arrow with its tail at the initial carbon content of the charge and its head at the final carbon content. Each arrow thus represents the amount and direction of the change in carbon content that occurred during equilibration.

For those experiments where no change occurred the corresponding equilibrium position is represented by a point. The constant $(pCO)^2/pCO_2$ curves were drawn between the arrows and through the points obtained by equilibration with the appropriate $(pCO)^2/pCO_2$ ratio. The equilibrium points shown for 0% of Cr were taken from the previously published data on the Fe–C system.

ERRORS

In the course of investigating the binary Fe–C system, an attempt was made to evaluate the principal sources of error inherent in the method used. These sources of error and the probable limits to which they might alter an individual equilibrium measurement, in the region of about 0·4% of C, which represented an average case, are listed in Table III.

For the Fe–C alloys, more than 50% of the experimental results fell within $\pm$ 2% of the equilibrium curves drawn. In view of this, of the data in Table III, and of the number of individual experiments involved, the error in the equilibrium curves due to known sources of random error can be taken as $\pm$ 2%. The maximum effect of sources of non-random error is apparently less than $\pm$ 1%. In practice, the accuracy of the data must vary with composition and temperature. It seems reasonable, however, to assign an overall probable error of $\pm$ 3% to the carbon values for the binary Fe–C alloys.

With the ternary Fe–Cr–C experiments, two further sources of error must be considered: the effect of nitrogen and the uncertainty in the chromium content.

The amount of nitrogen present in the gas mixtures never exceeded $\frac{1}{2}$ vol.-%. Thus, if S is the atomic percentage of nitrogen which, dissolved in a given alloy, would be in equilibrium with 1 atm. of nitrogen, by Sievert's law the amount of dissolved nitrogen in equilibrium with the gas mixtures used should never exceed $(S\sqrt{0{\cdot}005}) = 0{\cdot}071\ S$. From the available data[1, 6] on the solubility of nitrogen in Fe–Cr alloys, it was calculated that for the range of conditions studied in the present work there should never be more than 1 atom of nitrogen to 80 atoms of chromium in the melt.

It would seem unlikely that such a small proportion of nitrogen could significantly affect the carbon activity. To test the effect of nitrogen, and to indicate whether the data obtained for these alloys could be applied to normal industrial conditions, where the amounts of dissolved nitrogen may be appreciable, four experiments were carried out.

Nitrogen was passed into the mixing tube at various measured rates of flow so as to vary the nitrogen partial pressure in the gas mixtures from 0·005 to 0·375 atm. It was calculated that this would vary the atomic percentage of nitrogen in an alloy containing about 25% of Cr from about 0·1 to 0·8%. The partial pressures of CO and CO_2 were such as to maintain $(pCO)^2/pCO_2$ at 300/1. The initial carbon contents were chosen to ensure that equilibrium would

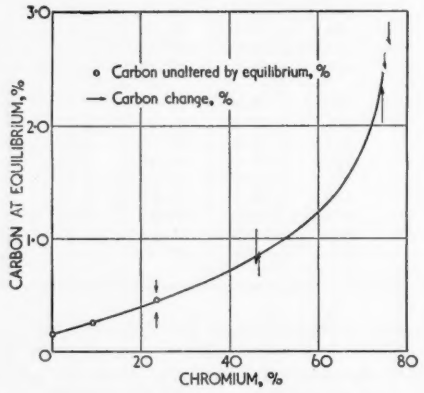

Fig. 3—Percentages of carbon in equilibrium with a $(pCO)^2/pCO_2$ ratio of 200/1 at 1760° C. as a function of chromium content

Table III

KNOWN SOURCES OF ERROR AND THEIR EFFECTS AT CARBON CONTENTS OF ABOUT 0·4%

Source	Effect on Measured Carbon (Probable Error), %
Random Errors	
Carbon analysis	$< \pm 2{\cdot}5$
$(pCO)^2/pCO_2$ measurement	$< \pm 1{\cdot}0$
Temperature measurement	$< \pm 1{\cdot}0$
Temperature fluctuation	$< \pm 0{\cdot}5$
Gaseous impurities	$< \pm 1{\cdot}0$
Total probable error	$< \pm 3$
Consistent Errors	
Thermal diffusion	$< +1$
Reaction with refractories	$< -\frac{1}{2}$
Reactions during quenching	$< -\frac{1}{2}$
Total probable error	$< \pm 1$

be reached within 2 hr. The equilibrium point was approached from the low-carbon side, because the effect of nitrogen would presumably be to raise the carbon activity and thus lower the percentage of carbon in equilibrium with the 300/1 CO/CO_2 ratio.

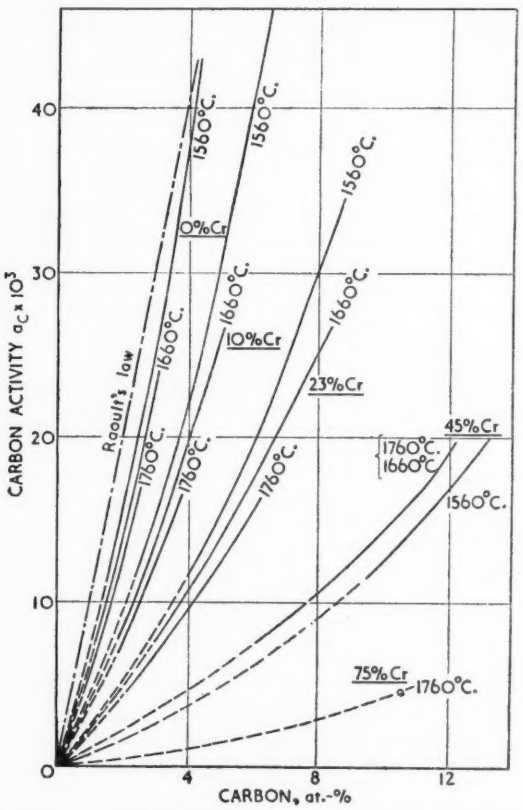

Fig. 4—Carbon activity relative to β-graphite as a function of atomic percentage of carbon at 1560°, 1660°, and 1760° C. for 0, 10, 23, 45, and 75% of Cr in the melt. Full lines show ranges of measurements; dotted lines are extrapolations

Table IV

EFFECT OF NITROGEN DISSOLVED IN A 25% Cr ALLOY ON THE CARBON CONTENT AT EQUILIBRIUM

$(pCO)^2/pCO_2 = 300/1$ at 1560° C.

p_{N_2}, atm.	N_2 in Alloy, at.-%	$(pCO)^2/pCO_2$	Time, hr.	Final Cr, %	Initial C, %	Final C, %
0·005	0·094	301	2·0	24·72	1·31	1·29
0·048	0·290	301	2·0	24·76	1·20	1·33
0·167	0·543	300	2·0	24·77	1·19	1·35
0·375	0·815	300	2·0	24·77	1·19	1·30

As may be seen from the results given in Table IV, the change in the equilibrium carbon content resulting from an eightfold increase in the number of dissolved nitrogen atoms lies well within the scatter of the results. The influence of small amounts of nitrogen dissolved in the melts may therefore be neglected.

The maximum error in estimating the loss of chromium during equilibration was $\pm 2\%$ of the final chromium content; this was for the longest time at 1760° C. The maximum error in the initial chromium content, due to segregation in the ingots from which the charge was made, was $\pm 1\%$ of the final chromium content. Thus, in the worst possible case, the error from these causes would be $\pm 3\%$ on the final chromium content. As this error was random and the number of experiments was large, its contribution to the probable error in carbon content can be safely taken as $< \pm 1\%$.

It therefore seems reasonable to assign an overall probable error of $< \pm 4\%$ to the equilibrium carbon values and to the derived activities and activity coefficients for the ternary melts.

DISCUSSION

The carbon activities as a function of the atomic percentage of carbon dissolved in the melt were derived from the curves in Figs. 2 and 3. They are shown in Fig. 4, for 0%, 10%, 23%, 45%, and 75% of Cr at 1560°, 1660°, and 1760° C. Pure β-graphite was taken as the standard state at each temperature, the activities being calculated from the relation

$$(pCO)^2/pCO_2 = K.a_{[C]} \quad \ldots\ldots\ldots\ldots\ldots\ldots (2)$$

the values of K, the equilibrium constant for reaction (1), being taken from the most recent compilation of the relevant data.[10] The full lines in Fig. 4 represent the activities in the range of the measurements; the dotted lines represent extrapolated values. In the case of the binary Fe–C alloys, the extrapolations are based on the equation used previously by the authors for this system.[8] This, in turn, was based on a simple statistical model for the Fe–C mixture. For such a model, Henry's law does not hold at any dilution; this can be seen more clearly in the activity-coefficient plot shown in Fig. 5.

From Fig. 4 it is evident that, for a given carbon content, the carbon activity is progressively decreased as the melts become richer in chromium. The carbon activity *decreases* slightly with temperature in melts

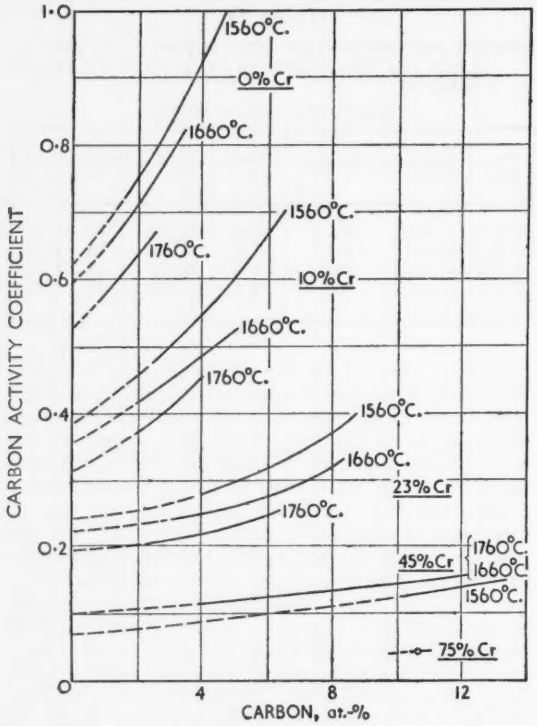

Fig. 5—Carbon activity coefficient ($\gamma_C = a_C$/atomic fraction C) relative to β-graphite as a function of atomic percentage of carbon at 1560°, 1660°, and 1760° C. for 0, 10, 23, 45, and 75% of Cr in the melt. Full lines show ranges of measurements; dotted lines are extrapolations

containing up to 23% of Cr, but *increases* with temperature in melts containing 45% of Cr.

In Fig. 5, the carbon activity coefficient is shown as a function of the atomic percentage of carbon dissolved in the different alloy melts for each of the temperatures studied. In the binary alloy, the effect of increasing the carbon content is to raise its activity coefficient markedly; *e.g.*, it is almost doubled by increasing the carbon content from 0 to 6 at.-%. This effect becomes progressively less with increasing chromium content until, at 45% of Cr, this increase in carbon content causes only a 25% increase in the activity coefficient.

The data from Fig. 5 are replotted in Figs. 6*a–c*, to show the activity coefficient as a function of the chromium content for constant carbon contents at each temperature. As would be expected, the rate $d\gamma_C/d[\%Cr]$ at which the addition of chromium lowers the activity coefficient decreases with temperature and with the amount of chromium present. The effect does not follow the simple type of relationship proposed by Gokcen and Chipman[4] to represent the dependence of the activity coefficient of oxygen in liquid iron on the concentrations of alloying elements such as aluminium, vanadium, or chromium.

By plotting $\log_{10} a_{[C]}$ against $1/T$ for each composition, it is possible, in principle, to derive the partial molar heats $\Delta\bar{H}$ and entropies $\Delta\bar{S}$ for the solution of pure β-graphite in these alloys. The accuracy of the data is not, however, sufficient to permit accurate values to be derived for these quantities. It can be shown, nevertheless, that $\Delta\bar{H}$, which is endothermic for carbon in pure iron ($+ 6{\cdot}4 \pm 2{\cdot}0$ kg.cal. per gramme-atom of β-graphite from 0 to 2·5 at.-% C), is little influenced by chromium concentrations up to 23%. The value of $\Delta\bar{H}_{[C]}$ at 10% and 23% of Cr

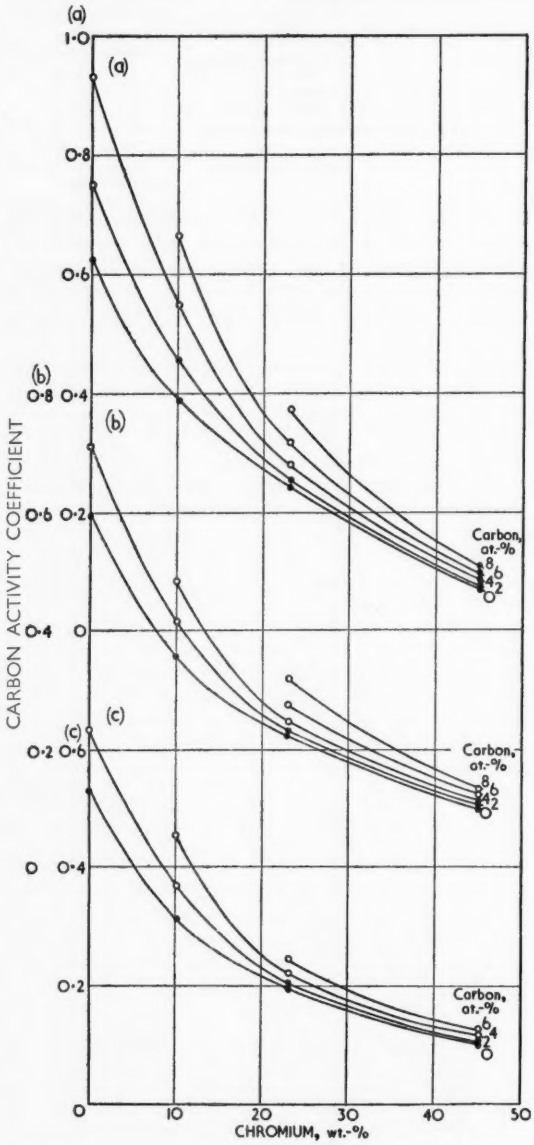

Fig. 6—Carbon activity coefficient at various carbon concentrations as a function of chromium content at (*a*) 1560°, (*b*) 1660°, and (*c*) 1760° C. Open circles represent points taken from the full lines in Fig. 5; solid circles are taken from the dotted extrapolations

is $8{\cdot}0 \pm 3{\cdot}0$ kg.cal. at 3 and 6 at.-% C, respectively. At 45% of Cr, however, $\Delta\bar{H}_{[C]}$ becomes exothermic by $-5{\cdot}3 \pm 3{\cdot}0$ kg.cal. at 10 at.-% C. In Fig. 4, at 45% of Cr, the activity lines for 1760° and 1660° C. overlap, whereas the line for 1560° C. lies below them. The line for 1660° C. should lie between those for 1560° and 1760° C., for it is unlikely that $\Delta\bar{H}_{[C]}$ could change markedly (in this case from 0 to -10 kg.cal.) over a 200° C. range. This error is indicative of the accuracy of the measurements and is consistent with the 4% errors proposed earlier for the measurements on the ternary systems. It is of interest to compare the $\Delta\bar{H}_{[C]}$ value of $-5{\cdot}3 \pm 3{\cdot}0$ kg.cal. with the values of $\Delta H^\circ_{1500^\circ\,K.}$ which can be calculated from the data cited elsewhere,[9] for the formation of solid chromium carbides:

$$C_{\beta\text{-gr.}} + \tfrac{3}{2}Cr_{(s)} = \tfrac{1}{2}Cr_3C_{2(s)} - 9960 \text{ cal.}$$

$$C_{\beta\text{-gr.}} + \tfrac{7}{3}Cr_{(s)} = \tfrac{1}{3}Cr_7C_{3(s)} - 14{,}150 \text{ cal.}$$

$$C_{\beta\text{-gr.}} + \tfrac{23}{6}Cr_{(s)} = \tfrac{1}{6}Cr_{23}C_{6(s)} - 17{,}300 \text{ cal.}$$

These values of ΔH are only weakly dependent on temperature. The heats of fusion of the carbides are probably little different from the heats of fusion of the various quantities of chromium of which they are composed, so that the values of $\Delta H^\circ_{1900^\circ\,K.}$ for the corresponding reactions for molten chromium and molten carbide should be about -10, -14, and -17 kg.cal., respectively.

The thermodynamic facts established in this work help towards the development of a structural model for Fe–Cr–C melts. The results already published by the authors on Fe–C melts[8] show that these may be pictured as a distribution of carbon atoms surrounded octahedrally by iron atoms, the distribution being such that there is a repulsive energy of about 3·6 kg.cal. between carbon atoms that are next nearest neighbours. This repulsive energy is the cause of the marked rise in the activity coefficient of carbon which occurs as its concentration is increased (*see* Fig. 5). This rise is markedly decreased by chromium, even at concentrations of 10 and 23%, so it would seem that chromium greatly decreases this repulsive interaction.

The most obvious structural model that can be proposed for an Fe–Cr–C melt is one in which the chromium and carbon atoms lower each other's chemical potentials by an attractive energy which causes them to cluster together. The distribution of carbon atoms relative to chromium should thus be non-random, as already proposed for oxygen and chromium in molten iron.[7] Each carbon atom should have, on the average, a higher proportion of chromium atoms among its nearest neighbours than exists in the remainder of the melt. It is of interest to note that the value of $\Delta\bar{S}_{[C]}$ that can be derived from the measurements at 45% of Cr and 10 at.-% C is $5{\cdot}8 \pm 2{\cdot}0$ cal./° C., compared with the value of $8{\cdot}5 \pm 2{\cdot}0$ cal./° C. that can be derived for 10 at.-% C in the absence of chromium, by extrapolation of the $\Delta\bar{S}_{[C]}$ values obtained for the binary alloys.[8] This lower value of $\Delta\bar{S}_{[C]}$ fits in with the clustering picture, but its accuracy is too low to corroborate it.

The effect of chromium on the behaviour of nitrogen in iron is somewhat similar to the effect on carbon described here. Data on the nitrogen solutions are available from the work of Brick and Creevy,[1] who measured the solubility of nitrogen at 1 atm. in various Fe–Cr melts at 50° C. above the liquidus, and at one other temperature for two compositions only. Figure 7 shows values for the solubility of nitrogen at a *constant temperature* of 1630° C., calculated from their results.* It appears that the effect of chromium on the nitrogen solubility increases greatly above 25%. This diagram may be compared directly with Fig. 3, from the authors' data, which shows the solubility of graphite at a fixed activity for the same range of alloys.

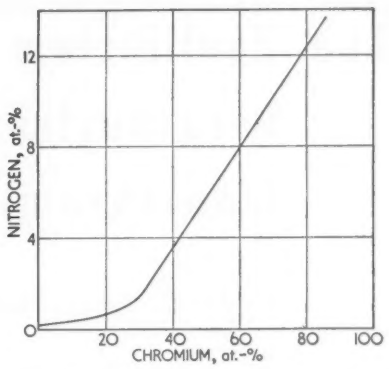

Fig. 7—Effect of chromium on the atomic percentage of nitrogen dissolved in liquid Fe–Cr alloys under 1 atm. of nitrogen at 1630° C. (calculated from the data of Brick and Creevy[1])

Acknowledgments

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* The results of Kootz,[5] who did not determine any temperature coefficients, are the same as those of Brick and Creevy up to 60% of Cr, but are rather greater between 60 and 90% of Cr.

The Equilibrium Controlling the Decarburization of Iron-Chromium-Carbon Melts

By W. E. Dennis and F. D. Richardson

SYNOPSIS

Data now available on the influence of chromium on the activity coefficient of carbon in iron–chromium alloys have been applied of the equilibrium, controlling the oxidation of carbon out of iron–chromium melts. This is shown to be

$$3[C] + Cr_2O_3 = 2[Cr] + 3CO.$$

The limiting carbon concentrations that can be reached for various chromium contents of the melt have been calculated and shown to agree with those obtained by Hilty by injection of oxygen gas into molten Fe–Cr–C alloys between 1711°and1826° C. The equilibrium conditions have been calculated for the wider temperature range 1550–1950° C. 794

In the production of low-carbon ferro-chrome and of low-carbon chromium steels it is often necessary to remove carbon from the metal by oxidation during refining. This removal is made difficult by the strong affinity of chromium for carbon; indeed, the extent to which carbon can be removed in normal practice without simultaneously oxidizing chromium is limited by the chromium content of the alloy.

It has been known for some time that preferential oxidation of the carbon is favoured by high temperatures. In 1948 this observation was put on a quantitative basis by Hilty,[3] who measured the amounts of carbon and chromium that remained dissolved in molten iron after injection of oxygen gas had produced a slag saturated with oxides of chromium. Hilty observed that the ratio of chromium to carbon remained approximately constant, and that the logarithm of this constant varied linearly with the reciprocal of the absolute temperature over the range 1711–1826° C. He therefore suggested that the decarburization equilibrium was*

$$(CrO) + [C] = [Cr] + CO \quad \ldots\ldots\ldots\ldots\ldots (1)$$

for which

$$K_1 = \% [Cr]/\% [C]$$

where $pCO = 1$ atm., and the activity of CrO was unity in the slag.

The adoption of this equilibrium carries with it the assumptions that (*a*) the activity coefficients of chromium and carbon do not vary with composition in liquid Fe–Cr–C alloys, and (*b*) the stable oxide of chromium above 1711° C. is CrO.

The present authors have recently shown that the influence of chromium on the activity coefficients of carbon in molten iron is large and markedly dependent on the concentrations of chromium and carbon in these melts.[5] The first assumption is thus untenable. The second assumption does not bear critical thermodynamic examination. If Hilty's equilibrium constant K_1 is combined with the well-established data for the oxidation of carbon,[6] it would appear that CrO should become increasingly more stable than Cr_2O_3 as the temperature is lowered. It has, however, been established that Cr_2O_3 is the stable oxide of chromium below 1400° C. Neither Boericke,[1] who studied the equilibrium

$$Cr_2O_3 + \tfrac{1}{6}Cr_{23}C_6 = 3CO + \tfrac{27}{6}Cr$$

up to 1400° C., nor Okada, Kokubo, and Matsuo,[4] who investigated the equilibrium

$$Cr_2O_3 + 3H_2 = 2Cr + 3H_2O$$

up to 1310° C., could find any evidence of chromous oxide.

Hilty recognized that the decarburization equilibrium might be

$$(Cr_2O_3) + 3[C] = 2[Cr] + 3CO \quad \ldots\ldots\ldots (2)$$

for which

$$K_2 = (a_{[Cr]})^2/(a_{[C]})^3$$
$$= (\gamma_{[Cr]}[\% Cr])^2/(\gamma_{[C]}[\% C])^3,$$

where a and γ are activities and activity coefficients. The explanation of his observations would then be that the activity coefficient of the carbon is lowered by the chromium in such a way that [% Cr]/[% C] remains almost constant. This was the explanation preferred by the present authors, and it was to test it that the study of the influence of chromium on the activity coefficient of carbon in liquid iron[5] was undertaken. The purpose of the present paper is to show that these new carbon activity coefficients can be used to predict the results of Hilty, and so establish that the controlling equilibrium is that given by

* Throughout this paper, brackets [] will be used to denote a constituent dissolved in the metallic phase and parentheses () to denote a constituent dissolved in the slag.

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Mr. Dennis was formerly with the Nuffield Research Group in Extraction Metallurgy (Royal School of Mines) of which Dr. Richardson is Director, and is now Senior Scientific Officer in the Ministry of Supply, Division of Atomic Energy (P), Culcheth, Warrington.

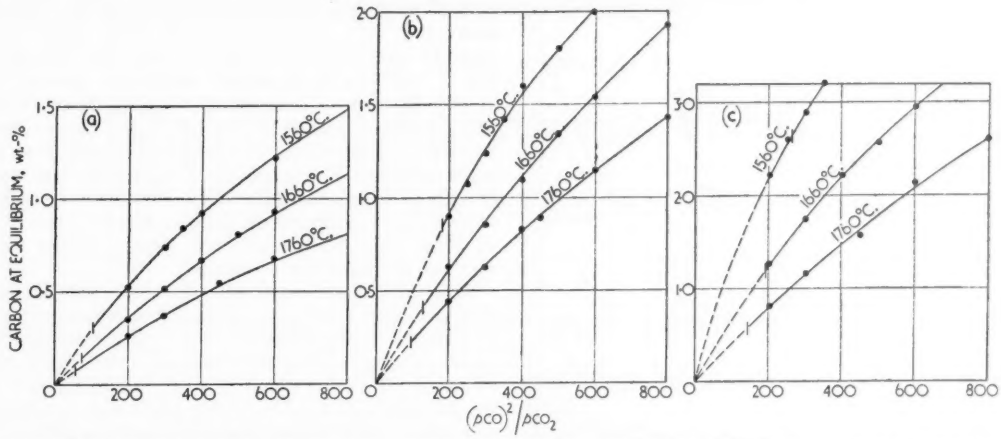

Fig. 1—Weight percentages of carbon in iron–chromium melts in equilibrium with various CO/CO_2 gas mixtures at 1560°, 1660°, and 1760° C., taken from Richardson and Dennis[5]: (*a*) 10% Cr–90% Fe; (*b*) 23% Cr–77% Fe; (*c*) 45% Cr–55% Fe. The vertical line cutting each isotherm shows the calculated value of $(pCO)^2/pCO_2$ in equilibrium with the alloy and free Cr_2O_3

equation (2). On the basis of this equation it is then possible to establish correctly how the Cr–C equilibrium is influenced by temperature.

THE DECARBURIZATION EQUILIBRIUM

The refiner of Fe–Cr alloys wishes to know the limits to which he can oxidize carbon out of his melts without removing chromium. He therefore wishes to know the chromium and carbon concentrations controlled by the equilibrium of equation (2). In this equilibrium, the CO pressure can be taken as 1 atm., for the amounts of CO_2 present are less than 0·5%; the Cr_2O_3 activity can be taken as unity.

To calculate the equilibrium carbon contents for these conditions, it is necessary first to calculate the values of $(pCO)^2/pCO_2$ for CO/CO_2 gas mixtures in equilibrium with various chromium contents and solid Cr_2O_3. This can be done by combining the data obtained by Chen and Chipman[2] on the equilibrium

$$2[Cr] + 3H_2O = Cr_2O_3 + H_2 \quad \ldots\ldots(3)$$

with the corresponding data[6] for the equilibrium

$$H_2 + CO_2 = H_2O + CO \quad \ldots\ldots(4)$$

The work of Chen and Chipman was confined to a single temperature (1595° C.), but they were able to calculate the effect of temperature on reaction (3) with some certainty from other data. Below 5·5% of Cr, however, $FeO.Cr_2O_3$ becomes the stable oxidation product, and data on equilibria involving it are available for 1595° C. only. For this reason, the following considerations do not apply to alloys containing less than 5·5% of Cr; these are, in any case, of little interest from a technical standpoint.

The percentages of chromium in iron, in equilibrium with Cr_2O_3 and a particular $(pCO)^2/pCO_2$ value for a given temperature, calculated in this way, apply strictly to pure Fe–Cr alloys containing no carbon. This is because the effect of the carbon present in the alloy on the activity coefficient of the chromium has not been taken into account. This effect should, however, be negligible at the low carbon concentrations of most interest to the refiner; it should be small over the whole range of carbon concentrations considered in this paper, as the atomic ratio of carbon to chromium never exceeds 1:5.

Figures 1*a–c* show the equilibrium carbon contents as a function of $(pCO)^2/pCO_2$ for melts containing various percentages of chromium at 1560°, 1660°, and 1760° C. On each isotherm a vertical line shows the calculated value of $(pCO)^2/pCO_2$ at which the chromium in the alloys is in equilibrium with free chromic oxide. The point of intersection of each vertical with its corresponding isotherm gives the carbon content of the alloy in equilibrium with Cr_2O_3. Figure 1*c* shows that, with a 45% Cr alloy at the lowest temperature (1560° C.), experiments were carried out at a $(pCO)^2/pCO_2$ value of 200/1 without formation of Cr_2O_3, whereas it was calculated that Cr_2O_3 should

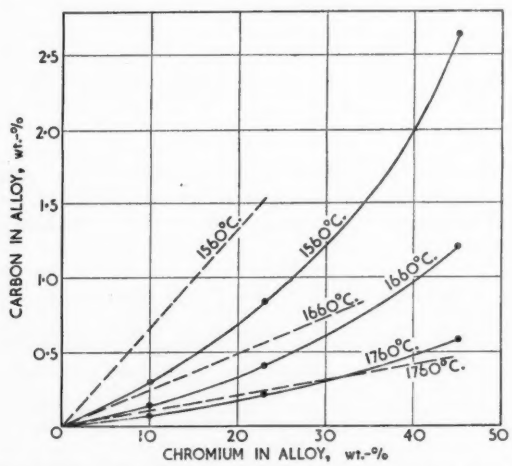

Fig. 2—Weight percentages of carbon in Fe–Cr alloys in equilibrium with free Cr_2O_3. Points are taken from Fig. 1*a–c*; broken lines represent Hilty's equation

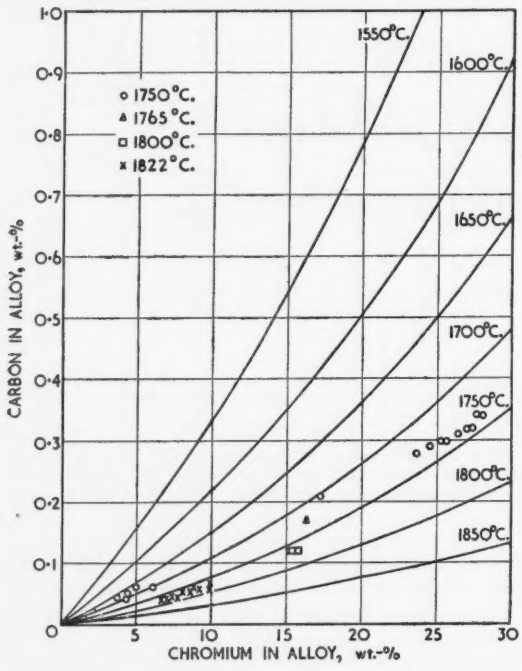

Fig. 3—Weight percentages of carbon in Fe–Cr alloys in equilibrium with free Cr_2O_3 at various temperatures. Points show Hilty's results with oxygen injection

be formed at all values of $(pCO)^2/pCO_2$ less than 250/1. Equilibration at values of $(pCO)^2/pCO_2$ lower than 200/1 produced Cr_2O_3, which was observed as a green film* on the quenched sample. At higher temperatures, the ratios at which Cr_2O_3 could be observed on the quenched heats of this alloy were in good agreement with the values calculated from Chen and Chipman's data.

These equilibrium carbon contents, taken from Figs. 1*a–c*, are shown in Fig. 2 as a function of the chromium content at 1560°, 1660°, and 1760° C. The curves drawn in Fig. 2 thus represent the levels to which the carbon content of a given alloy may be lowered without formation of free Cr_2O_3, for each of the temperatures studied. The linear relationships between % [Cr] and % [C] for the same three temperatures, calculated from Hilty's proposed equation

$$\log K_1 = \frac{15{,}200}{T} + 9{\cdot}46, \qquad \ldots\ldots\ldots\ldots\ldots\ldots (5)$$

are also shown in Fig. 2. It is evident that the carbon activities, together with Chen and Chipman's oxide equilibria, can be used to predict Hilty's results (*see also* Fig. 3). It is also clear that the extrapolation of Hilty's data to temperatures other than those he investigated (1711–1826° C.), on the basis of equation (1), leads to serious errors. By extrapolation and interpolation (on the basis of a straight-line log $a_{[C]}$ *vs.* $1/T$ plot), the relationships shown in Fig. 2 were calculated for temperatures between 1550° and 1850° C. at intervals of 50° C., and are plotted in Fig. 3. Hilty's experimental points are also shown in this diagram. The extent to which these experimental data depart from the calculated curves could be entirely due to the accumulated errors from the three separate investigations involved in the comparison. Indeed, the probable uncertainties in carbon analysis and temperature measurement in the work of Hilty and of the present authors could alone account for these discrepancies.

It is therefore concluded that the relationships shown in Fig. 3 will provide useful information about the effects of temperature and concentration upon the amounts of chromium lost during the decarburization of these alloys. Furthermore, it is considered that reaction (2) forms the correct basis upon which an understanding of these processes must be built.

It must be remembered that the present work and the measurements by Hilty were made with comparatively pure materials. Thus, the relations expressed in Fig. 3 apply strictly to 'impurity-free' melts. Although the effect of adding other elements on the thermodynamic properties of these melts has not been determined, except for nitrogen,[5] small proportions, not exceeding 1 in 20 atoms of any one of the constituents of impurities (other than strong carbide-formers, such as titanium or zirconium), should not cause any significant difference in the estimated equilibrium conditions.

Other things being equal, it may be stated that, from the viewpoint of reducing the chromium losses which occur during the removal of carbon by oxidation of these melts, the following conditions are desirable:

(1) A small slag volume and a slag in which the solubility of Cr_2O_3 is small

(2) High working temperature

(3) Reduced gas pressures (of CO and CO_2). The carbon activity is proportional to $(pCO)^2/pCO_2$ and thus to the total pressure; Cr_2O_3 activity is proportional to pCO_2/pCO and is thus independent of the total gas pressure

(4) The other elements present in the melt should be those which raise the carbon activity coefficient and/or lower the chromium activity coefficient.

Acknowledgments

The authors are indebted to the British Iron and Steel Research Association for financial support of this work and for a bursary to one of them (W.E.D.).

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* X-ray diffraction patterns obtained by Dr. B. R. Coles, of the Physics Department, for this substance were found to correspond to those for Cr_2O_3 and not to those for other possible oxides.

Sinter-Plant Assessment Trials at Dagenham and at Cleveland

By R. F. Jennings, B.Sc., E. W. Voice, B.Sc., A.Inst.P., P. K. Gledhill, Ph.D., G. C. Carter, B.Sc., and C. F. Ely

SYNOPSIS

Assessment trials have been carried out on two sinter plants: Ford Motor Co., Ltd., at Dagenham, and the Cleveland plant of Dorman, Long and Co., Ltd. Both plants used a variety of foreign ores. In both cases process variables were generally maintained very close to optimum levels, so that rates of sinter output and strand utilization were consistently high. The sintering air requirements mainly determined the outputs and varied from 42,000 to 89,000 cu. ft. S.T.P. per ton of raw mix, according to the constituents in the mix. The need was emphasized for eliminating obscured variations in the proportions and rate of delivery of raw material to the strand for optimum sintering conditions. Each plant has its own sinter cooling problem. Recommendations have been made towards solving these problems, at Dagenham by the addition of three extra windboxes, and at Cleveland by the use of a louvred cooling wharf. 814

In recent years the emphasis in research into sinter making has shifted from the problem of sinter quality to the assessment of plant performance in terms of output and its relation to plant design. Great expansion of sinter production is now in hand or projected in Britain, but there is considerable difficulty in predicting performance and output from past production data.

The United Steel Companies, Limited, recently made large-scale trials at the Appleby-Frodingham Steel Company* to provide additional operational data for the best use of existing plant and for the design of new equipment. These trials showed that the tonnage of material which could be sintered on a strand was directly related to the amount of air drawn through the bed. In fact, on sintering home ore, a specific quantity of air was required per ton of raw material. These findings emphasized that, with a given fan system, the permeability of the sinter bed strongly influenced the output from the machine as a higher permeability naturally permitted a greater rate of air flow.

Following this earlier work, it was thought useful to make similar trials on the performance of two Dwight-Lloyd sinter plants making sinter from a wide variety of foreign and home ore mixtures—in particular to determine the air requirements for these raw materials and to indicate how far outputs from the strands could be improved by improved permeability, mixing conditions, or varying proportions of return fines. The problem of cooling the sinter was also considered at these two plants, where quenching by water sprays was unsatisfactory for various reasons.

PLANT CONDITIONS AT DATE OF TRIALS

Sinter Plant at Ford Motor Co., Ltd., Dagenham (Fig. 1)

The sinter plant at Dagenham is a single Dwight-Lloyd strand, built by Huntington Heberlein and Co., in 1932. The dimensions are: length 52 ft. (divided into eight windboxes), width 6 ft., maximum bed height 12 in., grate area 306 sq. ft.

Ores, coke, flue dust, and cast-iron borings are delivered by transfer car into bins underneath an extension of the furnace high line. There is no screening or crushing plant; all materials are used as received. These raw materials are fed *via* rotating feed tables, from the bins, on to a gathering belt conveyor. Return-sinter fines, together with dust collected in the cyclone dustcatchers, are added to the raw mix as it passes up a second conveyor to a trommel mixer, which contains six integral lifting blades. Water is added to the mix when necessary by a hollow cone spray nozzle placed outside the end of the trommel. This spray is of a type used in oil burners, by which the water stream is broken up and accelerated by air pressure. From the trommel the feed falls directly down through a swinging spout against the cut-off plate, whence it rolls and slides on to the moving strand. No bedding layer is used.

The cut-off plate has a fixed angle of about 80° to the horizontal and has vertical side-pieces. The strand speed may be varied up to a maximum of

* E. W. Voice and others, *J. Iron Steel Inst.*, 1953, vol. 175, pp. 97–152.

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Mr. Voice is Assistant Director of the Central Research Establishment II, National Coal Board; the other authors are members of the staff of the N.E. Coast Laboratories of the Iron Making Division of B.I.S.R.A.

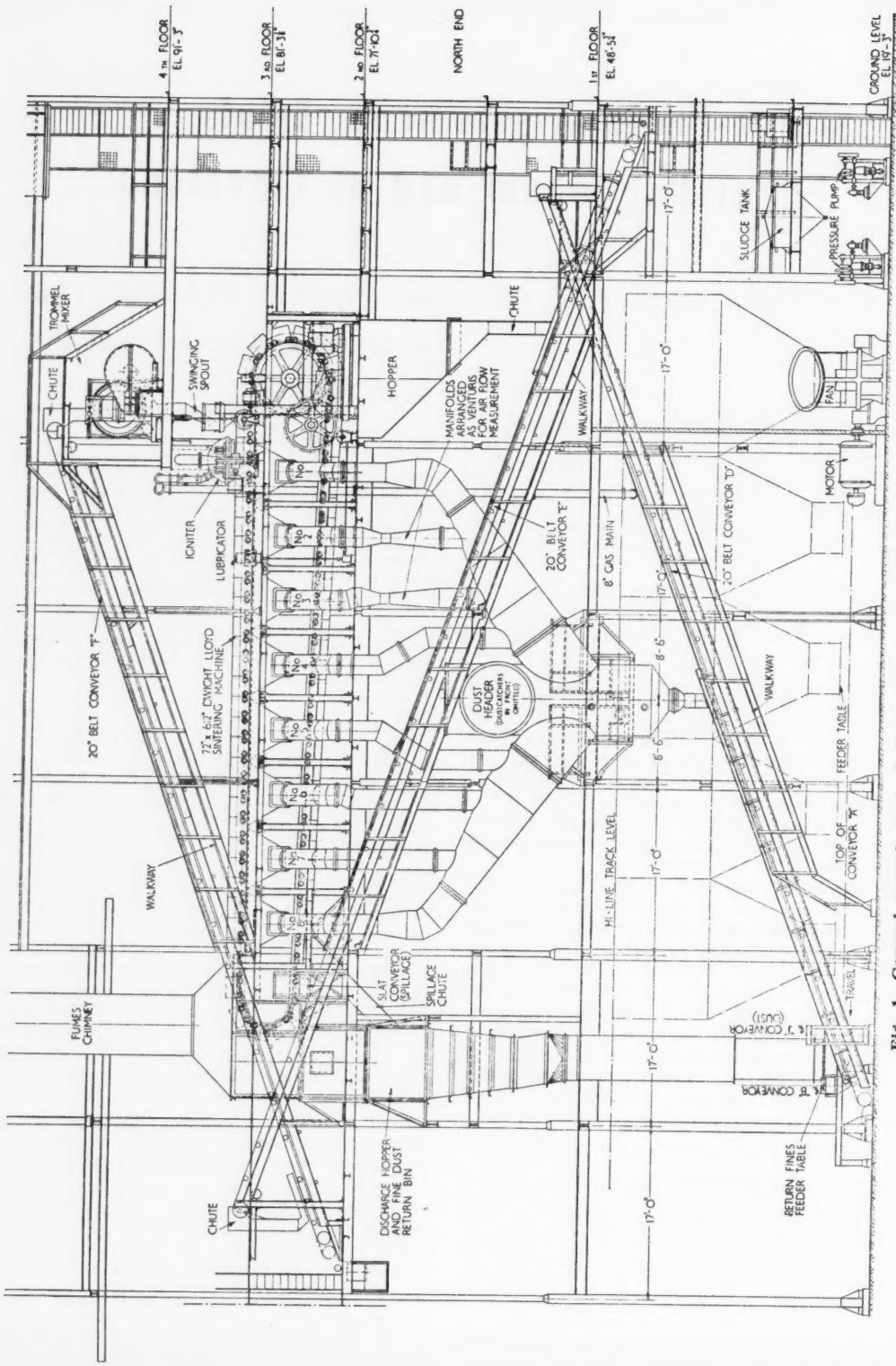

Fig. 1—General arrangement of sinter plant at Ford Motor Co., Ltd., Dagenham

5 ft./min. The grate bars are closely spaced to prevent losses of the fine ores and concentrates which form the bulk of the raw mix. Special 26% chrome steel fire bars are used and show excellent life. Ignition is by coke-oven gas, in which air for combustion is mixed with the gas before it enters the semi-circular ignition hood. The hood lies very close to the top of the bed; this is claimed to limit the amount of cold air drawn through the bed during ignition.

The pallets are pushed along the strand by the action of the head pulley and along lubricated slides which also act as air seals. At the end of the strand the sinter falls down a chute, over stationary bars $\frac{1}{2}$ in. apart, directly into a transfer car. The sinter is quenched by water sprays while in the car. Steam and hot gases are carried away in two chimneys, one outside the building, directly over the transfer car, and the other above the hood at the end of the strand.

The eight windlegs lead to two common mains, each sloping to join in a common dust header. This single main is divided into two cyclone dustcatchers and thence to the fan and chimney. The dust header and dustcatchers each have motor-operated outlet valves to ensure that suction is not lost when the dust is deposited on the uprising conveyor belts. Pressure drop through the dustcatchers is less than 1 in. W.G. at full fan rating.

The fan is a 16-bladed paddle type, driven by a 425-h.p. motor; it is rated at 90,000 cu. ft. S.T.P./min. at 1450 r.p.m., with 25-in. suction at an efficiency of about 70%, but is never run at more than 1000 r.p.m. The blades are refaced with hard weld metal after making 200,000 tons of sinter. As a result, the shape of the blades has somewhat altered during the past 20 years, and the original fan characteristics are probably not applicable to present operation.

Sinter Plant at Dorman, Long and Co., Ltd., Cleveland Works (Fig. 2)

The sinter plant at Cleveland is a single Dwight-Lloyd strand, built by Huntington Heberlein and Co., in 1949. The dimensions are: length 108 ft. (divided into 16 windboxes), width 6 ft., maximum bed height 13 in., grate area 648 sq. ft.

Raw materials are delivered by conveyor belt from the crusher house or by rail car into bins arranged in two lines. The ores are screened ($-\frac{1}{4}$ in.) at the screening house and the sinter is screened, after cooling, on $\frac{1}{16}$-in. vibrating-rod deck screens above the sinter despatch bin. Coke is crushed in rod mills to 70% $< -\frac{1}{16}$ in. before being fed into a bin. The raw-material bins use conventional feeder tables to deliver the mix on to two gathering belts.

A cross belt from both gathering belts delivers to an uprising main conveyor to the sinter plant. Some water is added to the mix on the first gathering belt after the addition of return fines. At the top of the main conveyor and above the sinter strand the mix passes over an inclined static screen of $\frac{3}{8}$-in. bar spacing. Any oversize materials passing the screens in the screening house are used as a bed layer. This material flows *via* a chute into a hopper in front of the swinging spout. This hopper is fitted with a spring-loaded front plate with the object of providing an even feed for the bedding layer. The remainder of the mix passes through the screen to a trommel mixer, 12 ft. 3 in. long and 6 ft. dia., rotated at $12\frac{1}{2}$ r.p.m.

The trommel is fitted with fixed paddle blades and water is added in a series of jets through a pipe extending two-thirds of the length of the drum with holes spaced along its length. There are no fine sprays. Provision is made for either salt- or fresh-water addition. Throughout the trials, only fresh water was used.

The mix is fed from the trommel to a short cross belt, from which it falls through the swinging spout on to the moving strand. Bed depths up to 13 in. are obtained by a variable cut-off plate; both the angle and the height of the plate are adjustable. A hood igniter is used, burning blast-furnace gas. Ordinary cast-iron grate bars were previously used and these are now being replaced by special 26% chrome steel bars. Strand speed is varied by the operator up to a maximum of 7·6 ft./min. The windlegs are connected to a common main. No cyclone dustcatchers are fitted. The dustlegs below the common main are emptied vertically on to a gathering belt through counterbalanced bell valves.

The fan is a double box-paddle type machine, driven by a 1050-h.p. motor. The motor is directly coupled to the fan, and the air volume can be adjusted by vane control. The designed rating of the fan is 125,000 cu. ft./min. at 300° F. against a static suction of 27 in. at 750 r.p.m.

Provision is made for water to spray on to the sinter bed over No. 15 windbox. This reduces the nuisance from dust at the end of the strand. The sinter is discharged down a chute to a breaker consisting of a rotary shaft with spaced claws. The sinter then drops into a despatch hopper, which is emptied periodically into a slag ladle. This ladle runs along a railway track at right-angles to the plant and is discharged at any desired point along the upper edge of a sloping cooling wharf. The latter is 240 ft × 40 ft., and its surface is set at an angle of 36°. The sinter is charged to a maximum depth of about 18 in. and remains on the bank for about 4 hr. before being discharged, through gates at the lower edge of the bank, on to a rubber gathering conveyor. Water has to be sprayed on to the sinter when discharging from the wharf, to aid natural cooling. The gathering belt is also sprayed. The sinter is then screened on a 1-in. bar grizzly and then through $\frac{5}{16}$-in. vibrating screens, the oversize passing into a bunker for despatch in wagons or belt for use in the blast-furnace, and the fine sinter goes to the return-fines bunker.

TRIAL CONDITIONS AND MEASUREMENTS

Trials at both plants were made under normal operating conditions with mixes determined by the ores to be handled in the plant. Measurements were made of the following quantities, to determine the sintering characteristics of the various mixes:

(1) Volume of air passing each windleg
(2) Air temperature in each windleg
(3) Rate of feed from the sinter tables (both average figures and variations)
(4) Water addition and proportion of moisture in mix

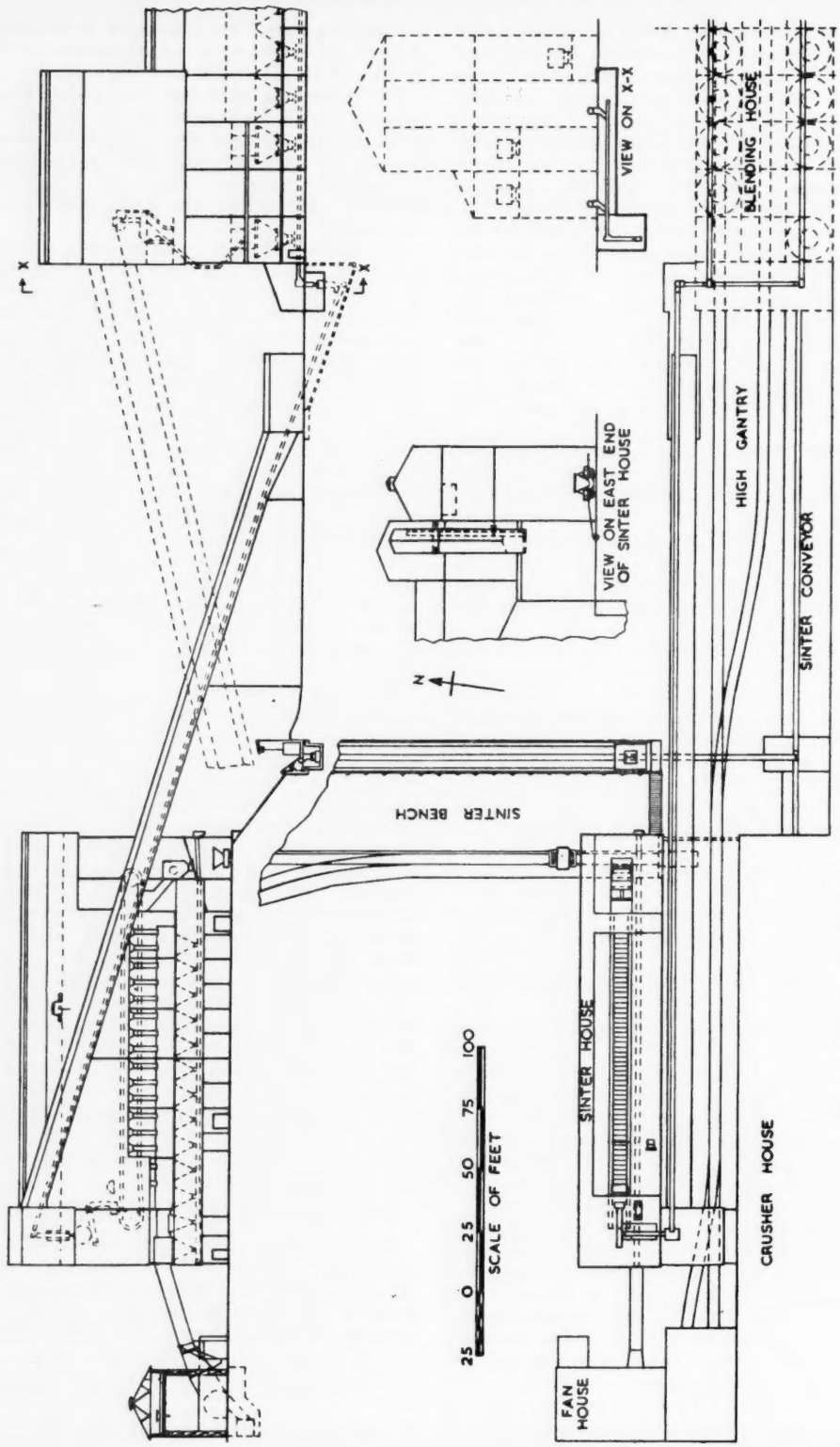

Fig. 2—General layout of the Cleveland sinter plant of Dorman, Long and Co., Ltd.

(5) Strand speed, bed height, and strand and fan suction (recorded from plant operational data).

Measurements were made with different types of raw mixes in each plant as follows:

Dagenham—

(1) Sierra Leone concentrates
(2) Sierra Leone concentrates plus a small proportion of Sierra Leone fines
(3) Sierra Leone concentrates plus a large proportion of Sierra Leone fines
(4) Sierra Leone fines plus Rif fines
(5) Rif fines
(6) Mixes (1), (2), and (3), with lime addition.

There was a small addition of cast-iron borings and flue dust in all mixes.

Cleveland—Mixes in this plant were very variable. The ores used were Cleveland, mixed Mediterranean, Swedish (Grangesburg and Gellivare *D*), Sierra Leone fines, Minera fines, and pyrites. The first four of these ores were present in most mixes. No flue dust was used during the trials.

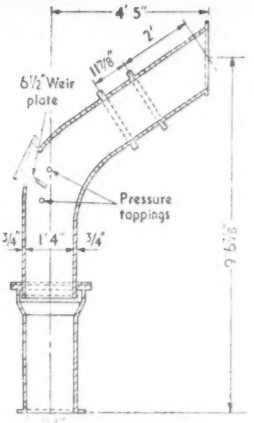

Fig. 3—Sectional elevation of windleg at Cleveland sinter plant, showing location of Weir plate for air-flow measurements

EXPERIMENTAL PROCEDURE

Air-Flow Measurement—The air volume flowing down the windlegs at Dagenham was measured by pitôt tube observation, as this plant is fitted with round windlegs. At Cleveland the air volume was measured by the pressure drop across a weir inserted in the rectangular windlegs (Fig. 3), this pressure drop being calibrated by reference to a ¼-scale model. Venturi-shaped restrictions were fitted in place of Nos. 2 and 3 windlegs at Dagenham and No. 4 windleg at Cleveland. A ring-balance gas-flow recorder was fitted to the Venturis towards the end of the trials at each plant, to give continuous air-flow measurement.

Table I

SUMMARIZED DATA ON DAGENHAM TEST

Date (1952):	25/3	26/3	27/3	28/3 a.m.	28/3 2–4 p.m.	28/3 4.30–4.40 p.m.	31/3 a.m.	31/3 1.45–2.30 p.m.	31/3 3–4 p.m.	1/4 a.m.	1/4 p.m.	2/4 2–2.30 p.m.	2/4 3–4 p.m.	7/4 a.m.	7/4 p.m.	8/4 p.m.
Bed height, in.	8	8	8	7¾	7¾	7¾	7¾	7¾	7¾	7¾	7¾	7¾	7¾	7¾	7¾	7¾
Strand speed (indicated), in./min.	48	48	48	49	49	49	44	41	42	46	47	44	44	56	56	48
Raw mix input rate, tons/hr.	46	46	42·9	42·9	42·9	43·4	40·3	36·7	37·2	41·2	41·2	39·2	40·2	50·3	50·3	46·5
Suction at windlegs, in. W.G.	19½	18	18	19	18	16½	18	17½	16	19	17	19	16	12	13	17
Hottest windbox number	7	7	...	7½	7	7½	8	7½	6½	7½	7½	7½	6½	7	7	6½
Maximum windbox temperature, ° C.	300	300	...	180	270	225	240	240	280*	240	240*	300	310	225	205	225
Sierra Leone conc. in mix, %	...	...	...	...	29·6	29·6	60·0	63·0	62·0	51·0	50·4	64·4	64·0	...	...	...
Sierra Leone fines in mix, %	40·2	40·2	40·0	40·0	39·8	39·8	10·5	...	...	...	...	...	...	...	...	39·0
Rif fines in mix, %	29·8	29·8	30·0	30·0	...	...	...	...	...	16·0	15·8	...	...	70·2	70·0	27·0
Return sinter in mix, %	19·5	19·5	10·1	10·1	14·0	14·0	14·0	19·0	18·8	17·0	16·5	19·3	17·9	14·0	14·0	14·0
Cast-iron borings in mix, %	2·6	2·6	2·6	2·6	2·3	2·3	5·2	4·0	4·0	3·6	3·6	3·1	3·0	3·0	3·3	4·6
Flue dust in mix, %	14·9	14·9	14·5	14·5	11·7	11·7	7·5	11·0	10·8	9·7	9·6	10·2	9·9	10·0	10·0	13·0
Coke in mix, %	3·0	3·0	2·8	2·8	2·6	2·6	2·8	3·0	3·1	2·7	2·9	2·0	2·0	2·8	2·7	2·4
Lime in mix, %	...	...	...	...	...	...	...	...	1·3	...	1·2	...	2·2	...	...	...
Total air flow at windlegs, 10^3 × cu. ft./min. (30 in. Hg, 60° F.)	55·0	†	†	57·1	68·7	66·6	55·0	66·6*	67·6	51·8	58·1	65·5	79·3	47·6	47·6	63·4
Sintering air flow (1st to hottest windbox inc.), 10^3 × cu. ft./min. (30 in. Hg, 60° F.)	42·0			50·4	50·7	61·6	55·0	59·4	45·1	46·8	55·2	58·5	55·1	35·2	39·9	41·4
Cooling air flow (hottest windbox to end of strand), 10^3 × cu. ft./min. (30 in. Hg, 60° F.)	13·0			6·7	18·0	5·0	0	7·2	22·5	5·0	2·9	6·0	24·2	12·4	7·7	22·0
Sintering air flow/ton raw mix, 10^3 × cu. ft. (30 in. Hg, 60° F.)	54·8			70·5	70·8	85·0	82·4	97·2	72·9	68·1	80·3	89·7	82·1	42·0	47·6	53·4
Cooling air flow/ton raw mix, 10^3 × cu. ft. (30 in. Hg, 60° F.)	17·0			9·4	25·2	7·0	...	11·8	36·2	7·3	4·2	9·2	36·1	14·8	9·2	28·4
Sintering zone permeability, B.P.U.	93			103	114	136	109	127	117	95	116	119	143	101	109	104
Cooling zone permeability, B.P.U.	202			206	285	167	...	232	256	155	190	200	275	250	148	240
Time to sinter, min.	11·2	11·2		11·7	10·9	11·7	13·8	14·1	11·8	12·6	12·9	12·9	11·2	9·5	9·5	10·4
Sinter output, tons/hr., from daily records	32·8	32·2	33·0	33·0	33·0	33·0	26·5	26·5	26·5	30·0	30·0	28·5	28·5	38·0	38·0	34·0

* Widely variable readings † Not measured

Windleg Temperatures—Windleg temperatures were measured by inserting sheathed copper–constantan thermocouples in the centre of the windleg.

Feeder-Table Rates—Feeder-table rates were measured by allowing the feed to fall on a tray placed on the gathering belt. The weight falling on this tray, together with a measured speed of the belt, gave spot measurement of the rate of feed. Water additions to the raw mix were measured in an integrating water meter, and moisture contents of the mix, before and after the mixer, were estimated from numerous spot samples checked in a Speedy Moisture tester and confirmed by drying tests in the laboratory.

CO_2 in Windlegs—At Dagenham, CO_2 measurements in the windlegs were obtained by dry gas samples taken at the point of pitôt tube insertion. These were later analysed in the laboratory.

General—Since a specified raw-material mix was rarely sustained for more than half a day, each test was considered to last 3 hr. During this period all the foregoing measurements were recorded at intervals of about ½ hr. Daily runs were repeated where raw material and blast-furnace demands made it possible. The techniques used in the trials were largely developed from those used during the sinter-plant trials at Appleby-Frodingham (*op. cit.*).

OBSERVATIONS AND RESULTS

The trials were carried out at Dagenham on nine working days between 25th March and 8th April, 1952. At Cleveland the trials were carried out on nine working days between 17th and 31st July, 1952.

Typical conditions observed during hourly periods in the trials are set out in Tables I and II, where the principal data are set out in similar form to those recorded in the trials at Appleby-Frodingham for direct comparison between both series.

From the plant observations the following derived data have also been measured, using the nomenclature developed at Appleby-Frodingham:

(*a*) Sintering air required per ton of raw mix,

$$V = \frac{F}{I} \text{ cu. ft. (S.T.P.)/ton}$$

(*b*) Sinter bed permeability during sintering and cooling on the strand,

$$P = \frac{F}{A} \times \left[\frac{h}{s}\right]^{0\cdot6} \text{B.P.U.},$$

as developed in recent work on the permeability of sinter beds.*

* E. W. Voice, S. H. Brooks, and P. K. Gledhill, *J. Iron Steel Inst.*, 1953, vol.174, pp. 136–139.

Table II

SUMMARIZED DATA ON CLEVELAND TEST

Date (1952):	17/7	18/7 a.m.	18/7 p.m.	21/7 a.m.	21/7 p.m.	23/7	24/7 3.10 p.m.	24/7 3.35 p.m.	25/7 a.m.	25/7 p.m.	28/7 1.15 p.m.	28/7 3.40 p.m.	30/7 a.m.	30/7 p.m.	31/7 1.45 p.m.	31/7 3.00 p.m.
Bed height, in.	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Stand speed (indicated), in./min.	80	79	74	70	79	80	82	72	86	83	83	74	88	78	70	56
Raw mix input rate, tons/hr.	113·9	97·8	103·2	100·6	113·1	125·5	110	103·8	98·4	98·8	88·2	115·3	116·3	109·4	86·6	87·5
Suction at windlegs, in. W.G.	21½	19½	22½	20½	20½	25	24	21	26	25½	23	22	24½	22	25	19
Hottest windbox number	15	14	16	16	15	16	15	16	16	16	16	15	16	16	16	14
Maximum windbox temperature, °C.	350	350	330	320	340	320	330	180	275	325	320	320	260	350	270	350
Cleveland in mix, %	7·9	10·2	17·1	10·4	9·4	8·6	11·1	9·8	9·1	9·5	10·6	8·1	11·5	10·6	7·3	11·1
Mixed Mediterranean in mix, %	11·9	13·9	15·1	...	14·8	13·4	13·8	15·7	17·2	17·1	18·3	13·0	15·8	17·1	30·4	12 8
Swedish in mix, %	29·1	26·6	24·4	30·8	28·6	21·5	27·0	22·3	30·5	28·5	31·9	24·4	23·9	17·5	19·5	30·0
Sierra Leone fines, %	9·2	5·8	6·3	9·6	4·8	4·8	2·7	4·3	5·1	...	4·7	3·9	3·2	3·4	8·3	8·2
No. 11 bunker, %	...	0·9	1·1	3·7	0·8	3·0	...	...	...	...	...	...	...	4·3	...	...
Minera, %	...	...	...	11·6	...	...	...	...	...	...	...	...	...	...	...	...
Return-sinter fines, %	35·2	36·8	28·9	27·0	33·7	41·8	37·1	39·7	28·6	37·2	25·7	44·0	38·7	39·1	26·0	30·0
Coke, %	4·0	2·9	4·7	4·7	4·5	3·7	4·3	5·0	4·2	4·7	5·5	4·2	4·5	4·3	5·4	4·3
Moisture in bed, %	†	6·5	6·5	7·1	7·0	7·3	†	†	9·6	7·4	6·5	7·0	5·2	6·9	6·8	7·5
Total air flow at windlegs, 10^3 × cu. ft./min. (30 in. Hg, 60° F.)	108·9	112·3	107·8	103·6	114·1	108·9	116·8	116·3	112·0	110·0	119·4	118·2	104·3	118·4	98·0	113·8
Sintering air flow (1st to hottest windbox inc.), 10^3 × cu. ft./min.	102·6	95·6	107·8	103·6	107·8	108·9	109·9	116·3	112·0	110·0	119·4	110·0	104·3	118·4	98·0	99·0
Cooling air flow (hottest windbox to end of strand), 10^3 × cu. ft./min.	6·3	16·7	...	...	6·3	...	7·1	...	...	...	...	8·2	...	...	...	14·8
Sintering air flow/ton raw mix, 10^3 × cu. ft. (30 in. Hg, 60° F.)	54·0	58·5	64·5	62·0	58·0	52·0	60·0	67·5	68·0	67·0	81·0	57·5	54·0	65·0	68·0	68·0
Cooling air flow/ton raw mix, 10^3 × cu. ft. (30 in. Hg, 60° F.)	3·3	10·2	...	...	3·4	...	4·3	...	...	...	...	4·3	...	...	...	10·1
Sintering zone permeability, B.P.U.	119	126	114	116	121	108	112	129	109	108	125	126	105	127	97	132
Cooling zone permeability, B.P.U.	110	154	...	...	115	...	117	...	...	...	...	141	...	...	...	140
Fan inlet suction, in. W.G.	23½	21½	24	22½	22½	27	26	23	28	27½	25	24	26½	24	27	21
Fan temp., °C.	138	166	121	121	138	115	132	138	115	115	115	138	121	132	138	154
Time to sinter, min.	14·8	14·0	17·0	18·0	14·7	15·8	14·4	17·5	14·6	15·2	14·5	15·7	14·3	16·3	18·1	19·4
Sinter output, tons/hr., from daily records	52·4	54·5	54·5	49·6	49·6	54·6	53·2	53·2	51·2	51·2	51·6	51·6	49·7	49·7	49·0	49·0

† Not measured

Air Required per Ton of Raw Mix

The assessment trials at Appleby-Frodingham had shown that, with a mixture of home ores, the average air required per ton of raw mix was 44,000 cu. ft. S.T.P., while laboratory experiments on sintering of foreign ores indicated that the air requirements varied according to the different ores composing the raw mix. The trials at Dagenham confirmed the laboratory experiments, and Table III (Nos. 1–7) shows the measured variation of air required with mixes composed of different ores. A raw mix of Sierra Leone concentrates, flue dust, return fines, coke, and cast-iron borings required about 90,000 cu. ft./ton (No. 1), but the introduction of Sierra Leone fines and Rif fines resulted in a progressive decrease in the air requirements (Nos. 3–5). A mix in which Rif fines was the only raw ore present required only 45,000 cu. ft./ton. An addition of only 2% of lime to a Sierra Leone concentrate mix also resulted in a reduction in air requirements (No. 2). Because of these variations, the output of a sinter plant is considerably influenced by the types of ore used, and for constant output the proportions of ores in the sinter mix must be carefully balanced.

The mixes used at Cleveland contained such a variety of ores in differing proportions that it was not possible to eliminate the influence of these variables on the air requirements per ton, which varied from 52,000 to 81,000 cu. ft. S.T.P./ton, with an average of 63,000 cu. ft. S.T.P./ton. The feeds were distinguished principally by the variations in the proportion of return fines, and the results indicated that the air required per ton of raw mix decreased as the proportion of return fines increased (Fig. 4). The variation in the types of ore used undoubtedly accounts for much of the scatter in the relationship.

Condition of Raw Material and Method of Feed to the Strand

There were considerable fluctuations in the rate of feed from the feeder table, and during both trials the fluctuations were greatest at the ore bins, where variations up to $\pm$ 25% of the average feed rate were observed. The coke feed was much steadier and at Cleveland the observed variation in coke feed was only $\pm$ 2%.

At Dagenham the wide variations in minute-to-minute feed rates at the bins were reflected directly on the strand. The material passed rapidly through

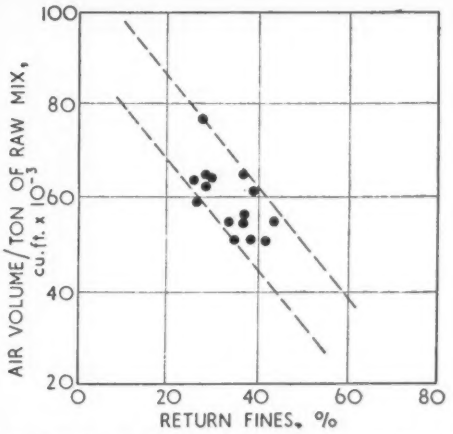

Fig. 4—Relation between percentage return fines and air per ton of raw mix at Cleveland

the trommel mixer, which could not act as a sufficient buffer against the variations in feed rate, and in consequence it was extremely difficult to maintain a constant level of material behind the cut-off plate. Water addition effectively maintained a flowing but coherent mass of material. The fixed angle of the cut-off plate did not always allow the material to roll on to the strand when material was piled against it, but when the level of the heap was kept low the material had to drop some 2 ft. from the swinging spout. This may have influenced the bed packing at such moments so that no increase in permeability was obtained. Since the swinging spout did not reach above the edges of the strand, this sometimes prevented the pallets from being evenly filled at the corners when the material was not heaped against the cut-off plate.

At Cleveland, since the rate of feed could not be controlled at strand level, accurate setting of the feeder tables was required to match the rate of sintering with the rate of feed to the strand. Variations in the feed from the feeder tables did necessitate continual changes in strand speed, but, because four ore bunkers were normally in operation, the variation in feed rate to the strand was much smaller than at Dagenham. Usually, a constant level of material was maintained behind the cut-off plate, despite the

Table III

VARIATION OF AIR REQUIRED WITH MIXES OF DIFFERENT ORES

No.	Raw Mix (ores only)	Proportion	Date	Air Flow, cu. ft./ton S.T.P.
1	Sierra Leone concentrates	Alone	2.4.52	90,000
2	Sierra Leone concentrates + 2% lime	Alone	2.4.52	82,000
3	Sierra Leone concentrates + Sierra Leone fines	5·7:1	31.3.52	82,000
4	Sierra Leone concentrates + Sierra Leone fines	3:4	28.3.52	77,000
5	Sierra Leone concentrates	3·2:1	1.4.52	68,000
6	Sierra Leone fines + Rif fines	1·4:1	8.4.52	53,000
7	Rif fines	Alone	7.4.52	45,000
8	Swedish, mixed Mediterranean, Cleveland, Sierra Leone fines	5:3:2:1	...	63,000 (average)

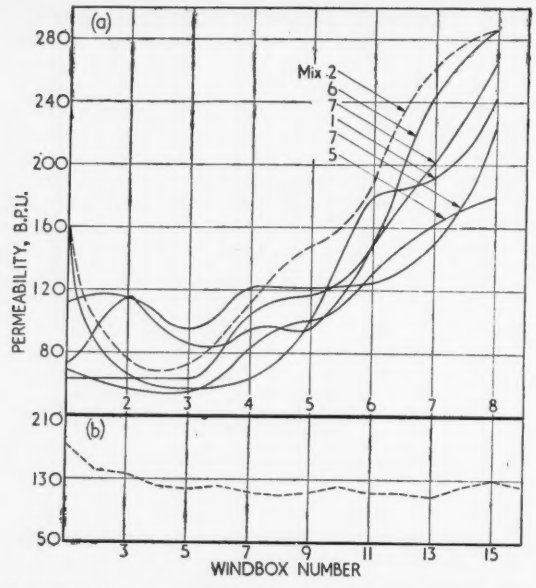

Fig. 5—Typical permeabilities measured along the strand at (*a*) Dagenham, (*b*) Cleveland

minute-to-minute variations from individual tables. The proportion of return fines could be reasonably well adjusted at the feeder table, provided that the hopper was kept full. However, when discharge from the cooling wharf was interrupted for some time, the feed rate could not be maintained since the sinter was screened after discharge from the wharf and under-size passed directly to the return-fines bin, which was relatively small.

A bedding layer was obtained by screening the mix on a static screen before the trommel mixer. The oversize material used for bedding consisted mainly of return fines. This screen often choked and the relative proportions of material going to the hearth layer, and through the mixer, were thus upset.

The swinging spout tended to pile the material against the cut-off plate slightly higher at the edges of the strand than at the centre, but with normal operation the material did not appear to pack against the cut-off plate, and changes in the method of feed from the swinging spout would not be expected to improve the permeability of the bed.

Permeability of the Sinter Bed

Typical curves showing the change of permeability with distance along the strand are plotted in Fig. 5*a* for the seven different mixes investigated at Dagenham, and in Fig. 5*b* for the Cleveland mix. Whilst, in general, the Dagenham mixes showed an initial decrease followed by a rapid increase in permeability along the strand, the Cleveland mix showed a minimum permeability about two-thirds along the strand and then only a slight increase in permeability to the end of the strand. The permeability at the end of the strand was generally lower than the sinter mix during ignition.

Typical variations of the air flow at No. 3 windleg, Dagenham, and at No. 4 windleg, Cleveland, are shown in Fig. 6. The fluctuations at Dagenham appeared to have a cyclic pattern, changes recurring three or four times per hour, which could have been caused by the variation in the return-fines feed rate. Although there was no evidence that higher permeability could be obtained by maintaining a low level of raw-material feed at the cut-off plate, abnormally high heaps of raw mix against the cut-off plate did reduce the initial permeability, particularly with Sierra Leone concentrates. The plant was operating

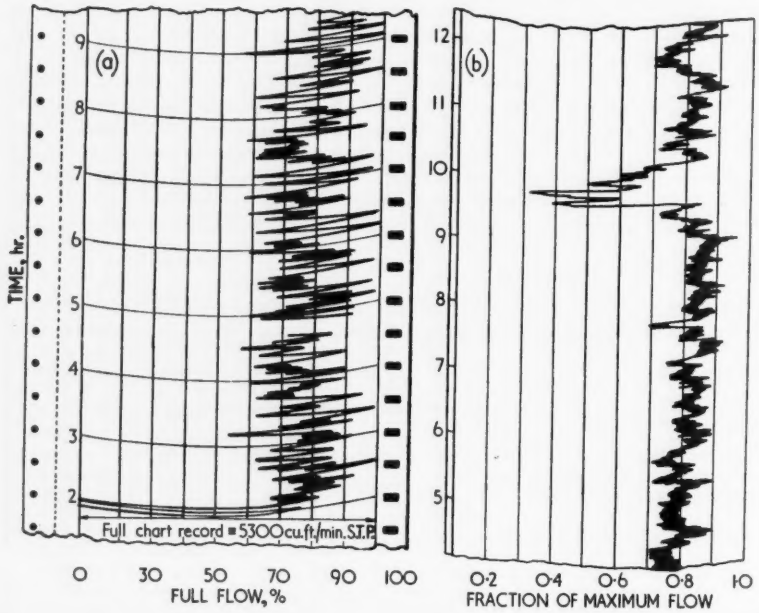

Fig. 6—Typical air-flow record charts: (*a*) No. 3 windleg at Dagenham; (*b*) No. 4 windleg at Cleveland

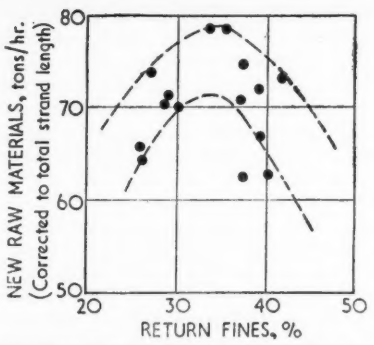

Fig. 7—Relation between hourly tonnage of new materials and percentage return fines at Cleveland

at its optimum moisture content in the mix and further additions reduced permeability.

The permeability of the Cleveland mix was increased when extra water was added, but difficulties were then experienced in igniting the mix and, under steady feed conditions, the operators did obtain the maximum permeability consistent with good ignition. Some lower permeabilities, by 20–30%, were caused by either temporary reduction in the return-fines feed rate or poor moisture control when the burden was charged.

Circulating Load

The trials at Appleby-Frodingham showed that the air requirements of raw mix, within the range investigated, were independent of the amount of return fines. The results at Cleveland indicated that the addition of return fines increases the efficiency of air utilization, so that, up to about 35% return fines, the decrease in air required per ton of raw mix is sufficient to offset the increased material to be sintered when using more return fines.

A possible reason for the different effect of return fines observed in the two practices is that, whilst return fines would be expected to be more easily sintered than the raw ores and hence should improve the air utilization efficiency, the air utilization with the Appleby-Frodingham mix is in any case twice as efficient as with the Cleveland mix, and thus the influence of return fines on the air required per ton

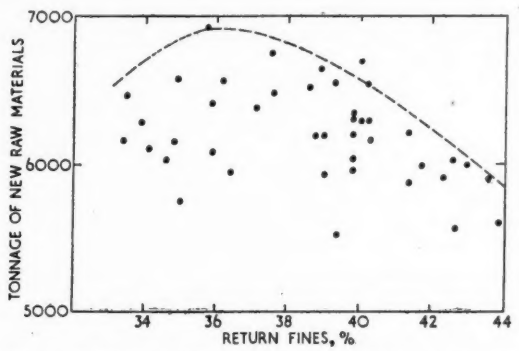

Fig. 8—Relation between weekly tonnages of new raw materials and percentage return fines at Cleveland

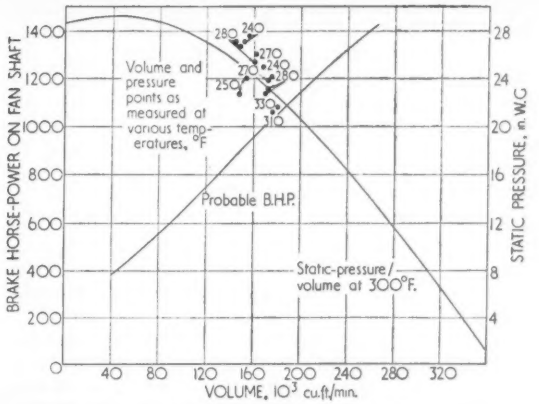

Fig. 9—Characteristics of fan system at Cleveland

of raw mix would be much more pronounced at Cleveland. An increase in blast-furnace sinter output can only be achieved if the quantity of ore (that is, new material excluding return fines) charged to the strand is increased; this is plotted against the percentage of return fines in Fig. 7. An optimum value of 36% return fines is indicated, although the points are somewhat scattered owing to variation in the components of the raw mix. Further confirmation is seen in Fig. 8, where the weekly tonnage of raw material sintered during 1952 is plotted against percentage of return fines.

At Dagenham the return-fines bin was directly below the sinter screens. It was therefore not possible to vary deliberately the proportion of return fines in the mix, the proportion used being that falling through the ½-in. static screens. When Sierra Leone concentrates were used without other ores, extra return fines (about 5% of mix) were added from one of the ore bins. These fines were obtained by grabbing material from the sinter stockyard.

Time to Sinter

The trials at both plants have enabled a comparison to be made between the sinter times obtained from similar sinter mixes on production plants and the B.I.S.R.A. sinter box. Using the formula (for notation, *see* paper on trials at Appleby-Frodingham)

$$t = \frac{VB}{26{,}900 \times P} \times \frac{h^{1\cdot6}}{s^{0\cdot6}}$$

differences in material and sintering conditions can be eliminated, and close agreement was found between actual and theoretical sintering times; *e.g.*, at Dagenham the sintering time for Sierra Leone concentrates was 14 min., compared with 38 min. in the B.I.S.R.A. box, *i.e.*, a ratio of 0·35, while the above formula gave a ratio of 0·36.

Air Utilization

In neither plant was the air volume drawn through the windlegs limited by the horse-power available at the fan, and vane or damper control was not used during steady operation. At Dagenham, however, it was difficult to correlate the measured air volumes and suctions with the expected fan characteristics.

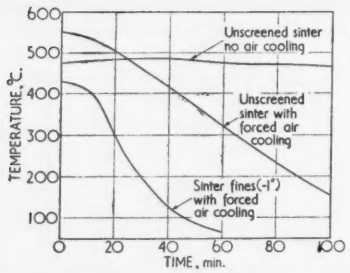

Fig. 10—Rates of sinter cooling on wharf at Cleveland

This was possibly due to the considerable modifications made to the fan blading during the last 20 years.

The air volumes recorded at Cleveland at various fan sections are in reasonable agreement with the characteristic curve of the plant fan supplied by the fan makers (Fig. 9), if allowance is made for pressure drop through ducting and the effect of the exhaust chimney draught—say, a total of 2 in. W.G. to be added to the strand suction. The air volumes measured in the windlegs do not allow for air infiltration through the dustleg seals, etc. However, a check estimate of total volumes, made by injecting a radon tracer sample at No. 16 windleg and sampling at the fan, agreed with these figures within the expected experimental error of $\pm 5\%$. Under average conditions the plant was using about 160,000 cu. ft./min. at 300° F., compared with a specified fan output of 125,000 cu. ft./min. at 300° F. The maximum air volume recorded was 181,000 cu. ft./min. at 330° F. and a suction of 21½ in. W.G. The metered electrical power used varied between 630 and 730 kW. per operating hour, *i.e.*, between 800 and 927 h.p. These figures show that the fan was usually running below its maximum power of 1050 b.h.p. However, this strongly suggests that the fan efficiency was above 60% over the whole range of air volumes recorded.

Table II shows that, at the Cleveland sinter plant, the strand utilization was 96%, and in most cases the hottest windbox was the last one. This value compares with the strand utilization of 84% determined at Appleby-Frodingham. In consequence, there was very little sinter cooling on the strand. At Dagenham the strand utilization was very variable and more difficult to assess accurately, as the machine was divided into only eight windboxes.

Sinter Cooling

The problem of sinter cooling was considered at both plants. At Dagenham, cooling is required to prevent failure of the concrete of the storage trough walls, and an average final sinter temperature less than 350° C. is required. Water-quenching may be destructive of sinter quality, and the constant drain of water from the sinter transfer cars was corroding the furnace high-line. Cooling of sinter by forced-air convection is the only alternative solution, and, because of space limitations, sinter cooling on the strand was the only method by which air-cooling could be used on this plant.

The amount of heat to be extracted from the sinter, after the assumed end of sintering, suggests that about 80,000 cu. ft. S.T.P. of air per ton must be used to cool the sinter adequately, compared with about 100,000 cu. ft./ton for full cooling (*see* paper on Appleby-Frodingham trials). If adequate sinter cooling is to be arranged on the existing strand the problem of matching the cooling air requirements to the strand suction and bed permeability must be appreciated.

Suppose that sinter cooling can be achieved by decreasing the sintering time, thus allowing a greater portion of the existing strand length to be used for cooling. Suppose also that 70,000 cu. ft. S.T.P. of air per ton of raw mix was required for sintering, and originally 20,000 cu. ft. S.T.P. of air per ton of sinter produced was available for cooling. For normal operating conditions, giving 40 tons/hr. of sinter (including return fines), 80,000 × 40/60 cu. ft. S.T.P./min. of cooling air will be required. The fan is therefore required to give an air flow of 106,800 cu. ft. S.T.P./min., compared with 66,800 cu. ft. S.T.P./min. for the assumed normal conditions. With a sintering zone permeability of 110 and a cooling zone permeability of 250, this would necessitate a suction under the bed of 39 in. W.G. To achieve this, the fan horse-power would have to be increased in the ratio:

$$\frac{39 \times 106{,}800}{18 \times 66{,}800} = 3{\cdot}5.$$

This would obviously be uneconomical.

A proposed solution was the extension of the strand by three windboxes attached to a separate fan for cooling. In this case 40,000 cu. ft. S.T.P./min. of air would have to be drawn through these windboxes. At a permeability of 250 this would require a suction under the bed of 13½ in. W.G., and hence the horse-power used for both sintering and cooling would only be increased in the ratio:

$$\frac{18 \times 66{,}800 + 40{,}000 \times 13\frac{1}{2}}{18 \times 66{,}800} = 1{\cdot}45.$$

If it had been possible to break up the sinter before cooling, the permeability of the bed would have been doubled, and with only three cooling windboxes the horse-power required for the cooling fan would have been only 14% of that of the sintering fan. Thus, the desired cooling of sinter on the existing strand would be prohibitive on power costs, but air-cooling using an extra three windboxes supplied with a separate fan would be possible, although still expensive in power costs.

At Cleveland the sinter cooling problem was very different. The sinter was required to be cool enough to be transported on rubber conveyor belts before screening. Little or no cooling takes place on the strand, since No. 16 windbox was usually the hottest and only rarely did the peak temperature appear at Nos. 15 or 14. The rate of cooling on the wharf was very slow, and temperature measurements of the sinter standing on the wharf surfaced with carbon blocks showed a temperature drop of only 10° C. in 2 hr. There was very little air circulation through the sinter mass, which was only cooled by natural convection and radiation from the surface. It was evident that the sinter was not cooled on the wharf

in any reasonable time, but the wharf served to even out the temperature of the sinter.

To prevent damage to the conveyors, substantial quantities of estuary water were sprayed on the sinter before discharging on the conveyor belt. This was harmful to its quality and introduced chlorides into the blast-furnace burden. One section of the wharf surface, about 7 ft. 6 in. × 18 ft., had been louvred to allow air to be blown through the sinter at a rate of about 11,000 cu. ft./min., with a pressure of 2 in. W.G. Much greater cooling was thereby achieved. The comparative rates of cooling observed on the wharf, with and without air flow, are shown in Fig. 10.

If the whole wharf surface were louvred for forced-air flow, then the cooling air for the whole of the present output of sinter could be supplied at the rate of 100,000 cu. ft./ton at low pressure and hence at low power cost.

CONCLUSIONS

Assessment trials both at Dagenham and at Cleveland have shown that process variables were generally maintained very close to optimum levels, so that rates of sinter output and strand utilization were consistently high. The outputs obtained were mainly determined by the air requirements for sintering. This requirement varied, according to the constituents of the raw-material mix, from about 42,000 up to 89,000 cu. ft. S.T.P. per ton of raw mix.

These air requirements were apparently independent of the rate of air flow and the permeability of the sintering zone. Fluctuations in average permeability in the sintering zone were considerable, reflecting minute-to-minute variations in the condition of the raw material on the strand. This emphasizes the need for eliminating obscured variations in the proportions and rate of delivery of raw material to the strand for optimum sintering conditions.

With certain sinter mixes the proportion of return fines can influence both the sintering zone permeability and the air required per ton of raw mix. Under such conditions the optimum proportion of return fines for maximum output of blast-furnace sinter has been as high as 35% of the raw mix.

The sinter cooling problems at each plant were widely different. Calculations of cooling zone permeability and cooling air requirements have been made to match suggested solutions using forced-air convection, by the addition of three extra windboxes at Dagenham and the use of a louvred cooling wharf at Cleveland.

Acknowledgments

The authors wish to thank the managements of the Ford Motor Co., Ltd., and of Dorman, Long and Co., Ltd., for permission to carry out these trials and to publish this paper. Particular thanks are due to Mr. F. Pearson, Manager of Blast Furnaces and Coke Ovens, Mr. D. W. E. Lloyd, Blast Furnace Superintendent, and his assistant, Mr. P. O. Sullivan, at Dagenham; and to Mr. W. Bell, Ore Preparation Plant Manager, and his assistant, Mr. K. Lewis, at Cleveland.

Effect of Mineral Additions and Moisture Control on the Sintering of Sierra Leone Concentrates

By P. K. Gledhill, Ph.D., G. C. Carter, and C. F. Ely

SYNOPSIS

The addition of lime and lime-bearing materials to Sierra Leone concentrates resulted in a decrease in sintering time and a reduction in air requirements. There was no appreciable change in permeability. With 4% of lime the air utilization efficiency was almost doubled. With concentrates of abnormally small particle size, permeability was appreciably greater at 6½–7% moisture, and sintering times were lowest at 5% moisture. Additions of blast-furnace slag considerably reduced sintering times at all moisture contents. 813

THE use of a high proportion of Sierra Leone concentrates in sinter-plant mixes results in a low plant output because such mixes require more air per ton to sinter than most other mixes and are much less permeable, so that air can be drawn only slowly through the sinter bed. Several authors[1] have stated that greater sinter production can be obtained by incorporating lime in sinter mixes, and the effect of lime and lime-bearing materials on the sintering of Sierra Leone concentrates on pilot plant scale was therefore studied.

After the first series of experiments was completed some concentrates of abnormally small particle size proved difficult to sinter on a Greenawalt plant. The effect of mineral addition and moisture percentage on the sintering characteristics of these fine concentrate mixes was therefore investigated. A crushed blast-furnace slag was used, since this material was readily available at the plant concerned and was

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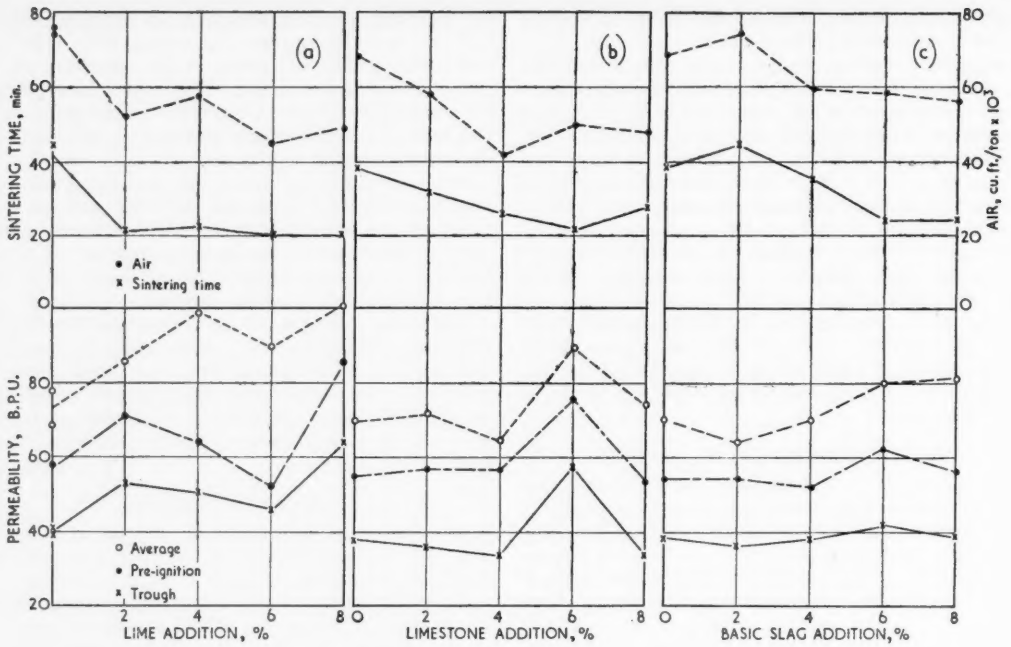


Raw Mix	Proportions
Normal Sierra Leone concentrates	64
Coke	6
Return fines	30
Lime addition (Fig. 1*a* only)	0–8
Limestone addition (Fig. 1*b* only)	0–8
Basic slag addition (Fig. 1*c* only)	0–8

Fig. 1—Effects of additions of (*a*) lime, (*b*) limestone, and (*c*) basic slag, on sintering characteristics of normal Sierra Leone concentrates

normally added to the blast-furnace burden to increase the slag volume.

EXPERIMENTAL PROCEDURE

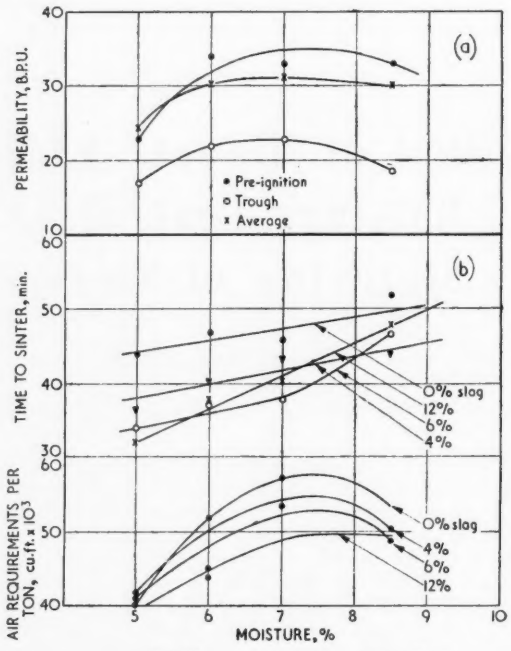
Experiments were carried out on the B.I.S.R.A. experimental sinter box,[2] with a constant bed height of 10 in. In the first series of experiments, using normal Sierra Leone concentrates, 30% return fines and 6% coke were used at a suction of 16 in. W.G. The moisture content of the mix was constant at 8%, which gave the maximum pre-ignition permeability. The effect of additions of up to 8% of fine slaked lime (containing 61% free lime), limestone, and basic open-hearth slag were investigated, and the sintering time, permeability,[3] and air required per ton of raw mix[4] were calculated.

In the second series of experiments, with fine Sierra Leone concentrates, 15% return fines, 6% coke, and 27 in. W.G. suction were used to correspond to Greenawalt plant conditions. The effect of additions of up to 12% of blast-furnace slag, crushed to $-\frac{3}{16}$ in. and containing about 60% $-\frac{1}{8}$-in., was investigated at moisture contents of the mix from 5 to 8½%.

RESULTS

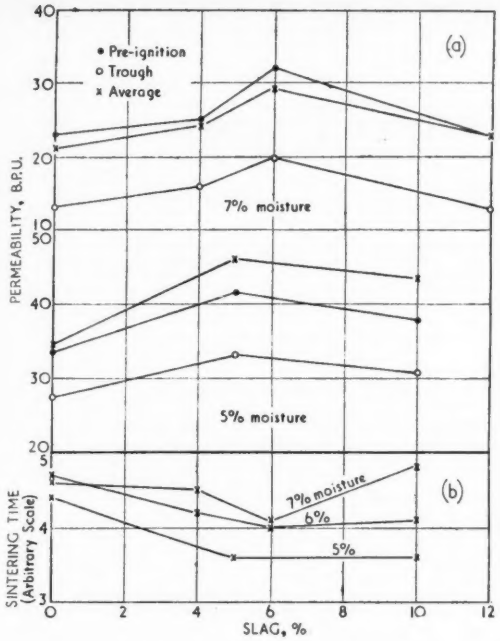
Normal Concentrates

The effects of lime on the sintering characteristics of normal Sierra Leone concentrates are shown in Fig. 1*a*. Sintering time decreased progressively as the



Raw Mix	Proportions
Fine Sierra Leone concentrates	79
Coke	6
Return fines	15
Moisture	5–8½
Blast-furnace slag (Fig. 2*b* only)	0–12

Fig. 2—(*a*) Effect of moisture content on permeability of fine concentrate mixes containing no slag. (*b*) Sintering times and air requirements of mixes containing varying proportions of moisture and slag addition



Raw Mix	Proportions Fig. 3*a*	Fig. 3*b*
Fine Sierra Leone concentrates	79	79
Coke	6	6
Return fines	15	15
Blast-furnace slag	0–12	0–10
Moisture	5%, 7%	5%, 6%, 7%

Fig. 3—(*a*) Effect of slag addition on permeability at moisture contents of 5 and 7%. (*b*) Calculated sintering times for constant weight of iron units

amount of lime was increased, although the permeability of the mixes was not appreciably changed. However, the air requirements of the mix were reduced and the efficiency of utilization of air was almost doubled with lime additions of 4% and more.

Limestone and basic-slag additions also reduced the air requirements of the mix and the sintering time, as shown in Figs. 1*b* and *c*, but the effect was not so pronounced as with lime. The experiments suggested that the beneficial effect was due to the amount of free lime present and that the limestone had a reduced effect because it must first be calcined in the sintering zone before affecting the sintering process.

Fine Concentrates

The size analyses of the normal and fine concentrates are compared in Table I, which shows that the material below 150-mesh in the fine concentrates was more than double the normal amount. Figure 2*a* shows the effect of moisture content on the permeability of the fine concentrate mixes containing no slag. Permeability was appreciably greater in the mixes containing 6½–7% moisture, but minimum sintering times were obtained with mixes containing 5% moisture.

Figure 2*b* shows the sintering times and air requirements of the mixes containing varying proportions

Table I

SIZE ANALYSES OF SIERRA LEONE CONCENTRATES

Screen	Normal, %	Fine, %
+ 60	35·3	15·2
− 60 + 85	23·3	16·1
− 85 + 100	10·5	11·6
− 100 + 120	10·4	16·1
− 120 + 150	7·7	14·8
− 150 + 200	6·2	11·6
− 200	6·3 (12·5)	14·5 (26·1)

of moisture and slag addition. These results demonstrate that the moisture content of the mix controlled its air requirements, and that the reduced air requirements of the mix containing 5% moisture more than offset its lower permeability and gave the shortest sintering time. This reduction in air requirements with reduction in moisture content was also observed when slag was added to the mix, and with these fine materials the achievement of greatest permeability by moisture control did not result in faster sintering.

The addition of slag considerably reduced the sintering time at all moisture contents, since it had the double rôle, in additions of up to 6%, of reducing the air requirements of the mix and increasing the average permeability. The effect of slag on the air requirements per ton of raw mix is shown in Fig. 2*b*, and its effect on permeability at moisture contents of 5 and 7% in Fig. 3*a*. The addition of slag reduces the iron content of the mix, and sintering times have been corrected in Fig. 3*b* to show the relative times of sintering mixes with a constant weight of iron units. This shows that, of the fine concentrate mixes investigated, the best conditions for sintering were obtained with 5% moisture and 5% slag addition.

CONCLUSIONS

The results show that the addition of lime-bearing minerals to a mix composed wholly of Sierra Leone concentrates and return fines can substantially reduce sintering times. This reduction is achieved principally by a reduction in the amount of air required to complete sintering, and possibly to a very minor extent by improved permeability.

The reduction in sintering times obtained by up to 6% mineral addition to a mix containing fine Sierra Leone concentrates more than offsets the resulting reduced iron content of the sinter.

To obtain full advantage of such mineral additions it is equally important also to exercise close moisture control.

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Sintering as a Physical Process

By H. B. Wendeborn

SYNOPSIS

The physics of sintering processes are described in the paper in a comprehensive and practicable form, to help blast-furnace and sintering-plant engineers to consider their problems in a co-ordinated way. In the course of sintering the ores are successively exposed to different temperatures, so that a sintering charge is divided into a number of parallel zones (called 'differential layers'), each of which has a different temperature. The travelling speeds of these temperature zones within the charge are a function of the fuel content and grain size, the reaction work to be performed, the gas resistance of the ore at the different temperatures, and the speed of the air flow through the charge. The temperatures created during sintering are an integral function of the heating sequence in the charge strata, during which the addition of the heat of combustion in the upper strata to that in the lower strata causes a typical heat accumulation. The function of the air current is therefore mainly that of a heat-conveying medium; part only of the air current effects in addition an intensified combustion of the preheated fuel. Measuring and plotting against time the instantaneous values of the waste-gas temperature (and especially the temperatures at a point within the charge) offer a convenient way of observing the behaviour of the ores, as well as the efficacy of the measures taken, and for checking the thermal intensity of the process. Examples of a heat balance are given.

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UNTIL a few years ago the greatest attention was paid in ironworks to the operation of the blast-furnaces; any existing sintering plants were treated as secondary, since they merely worked in conjunction with the main plant and the sintering process appeared to be simple. For this reason, previous publications dealing with the more theoretical bases of the sintering processes received very little attention in practice. The present paper is intended to give a description of sintering considered as a physical process and especially as a heat process.[1]

THE DOWN-DRAUGHT SINTER PROCESS

The ore fines for sintering are mixed with fuel, moistened, and loosely fed on to a grate in a layer 10–35 cm. thick, the grate being static or, in most cases, that of a continuous sintering machine. Combustion is initiated by igniting the fuel on the surface of the ore, and it is maintained by sucking air through the charge; combustion proceeds in the direction of the gas stream and effects a sintering or agglomeration of the charge.

The combustion process in this case does not take place simultaneously through the entire thickness of the layer, but proceeds as a thin horizontal layer moving vertically through the charge. This is called the 'differential layer,' the thickness of which corresponds to only a fraction of the total thickness of the charge layer (*see* Fig. 1). In this way are effected those heat processes which take place during down-draught sintering, the investigation and knowledge of which is the basis for the understanding and therefore the economic utilization of this process. A prerequisite for its performance is good gas permeability of the charge. Ore fines and dust-like raw materials are made moist and crumbly by adding water to them in mixing machines, so as to make them easily permeable by the gas. The gas permeability of such a layer 10–35 cm. thick spread on a suction grate is about 1·5 cu. m./sec. per sq. m. of suction surface. The technique for controlling it by measuring the air quantities at the individual suction boxes has been investigated recently by Voice, Brooks, and Gledhill.[2]

However, it is not sufficient for the ores to have a good gas permeability in a moistened state; they must also retain this condition after drying, as well as in the state of softening and at the beginning of melting.

The thermal course of the sintering process is illustrated in Fig. 1, from which any particular point in the process may be established. Above the combustion zone there is a layer of highly heated sinter material that is cooled by fresh air which, preheated in this way, arrives at combustion zone 1; below layer 1 the heat content of the waste gases is absorbed almost completely by the lower layers 2 and 3, so that they are preheated; in addition the hot waste gases cause the water in the lower layers to evaporate. Combustion air and charge meet in a highly preheated state in the combustion zone and produce such a high temperature there that in most cases an immediate melting of the material takes place. This high thermal efficiency is therefore produced by a heat accumulation within a partial layer of the charge, the 'sintering zone.' This sintering zone travels at 1–3 cm./min. towards the grate. At first the heat content of the waste gases of the lower layers is completely absorbed, on account of the large surface of the fine ore particles; later, as the combustion zone approaches the grate,

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more and more heat is carried away by the waste gases. When the combustion zone reaches the grate, a substantial portion of the total heat has already escaped with the waste gas. If the suction of air through the sintering material is continued until the latter is thoroughly cooled, all the liberated heat (with the exception of insignificant losses) can be carried away with the waste gas.

The combustion process with down-draught sintering therefore has a character entirely different from a conventional combustion process, because heat transmission within the charge is effected largely by the gas passing through; owing to the large interior surface of the particles of the material, a large portion of the waste-gas heat and the combustion heat is utilized directly for the process itself.

GAS AS HEAT TRANSMITTER

It is conceivable, for example, that with a small-grain-size charge which contains no fuel, and the surface of which has been preheated, the heat content of the surface of the charge may be made to travel in the flow direction of a gas that is sucked through. It is not, however, possible in this case to transfer the initial temperature of the surface to the lower layers, since, owing to the alternate heat transfer from the solid to the gaseous material and *vice versa*, a distribution of the heat on to both materials and a consequent decrease in temperature are unavoidable. To maintain or produce a certain temperature for every differential layer, the addition of a specific, though small, quantity of heat is therefore necessary to compensate for the loss in temperature; *i.e.*, the charge must contain fuel.

To analyse these conditions approximately by computation, it is assumed that one portion x of the heat is transferred indirectly by the gas, and another portion y is transferred directly by conduction and radiation from one differential layer to the next. The ratio of x to y is dependent on the nature of the charge, and on the speed of the gas flow.

For this example, in a first approach to the synthesis of the down-draught suction process, the process is assumed to be divided into two stages. For the first stage half the heat content of a partial layer a is assumed to be transmitted directly to the adjoining partial layer b by conduction and radiation.

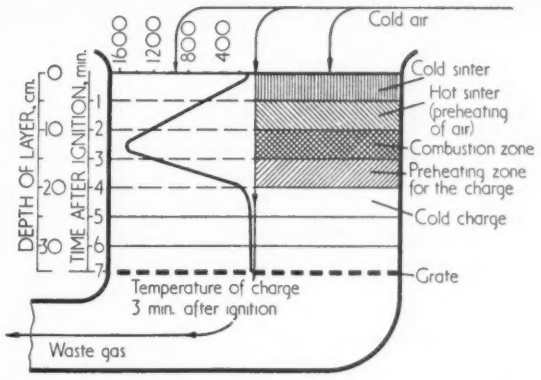

Fig. 1—Temperatures prevailing in a sinter 3 min. after ignition

For the second stage it is assumed that (after the heat transmission according to stage 1) a portion of the heat content remaining in the differential layer a is transmitted to the adjoining partial layer b by means of gas. These schematic calculations are based, *inter alia*, on the assumptions of complete heat exchange between partial layer and air and the substitution of heat-equivalent quantities for solid materials and air.

Initially, the first partial layer and the first quantity of air have a temperature of 1000° C., and all other partial layers and air quantities are at 0° C. The temperatures of the different partial layers at different times are shown in Table I, which is calculated on the basis of the following schedule.

At the beginning (stage 1), half the heat content of layer a travels from the initial temperature of 1000° C. by direct conduction and radiation into layer b. Both layers therefore have a temperature of 500° C. Then (stage 2) the air from layer a flows into layer b with an initial temperature of 1000° C.

Table I

DISTRIBUTION OF TEMPERATURES DURING SINTERING

Layer	Quantity of Air					
	0	1	2	3	4	5
	Temperature, ° C.					
a	1000	250	62·5	15·6	3·9	0·95
b	0	$(750 + q) = 1000$	437·5	156·2	50·8	15·6
c	...	0	$(750 + q) = 1000$	578·1	261·7	102·5
d	...	...	0	$(750 + q) = 1000$	683·8	367·2
e	...	...	...	0	$(750 + q) = 1000$	762·8
f	...	...	...	...	0	$(750 + q) = 1000$
g	...	...	...	...	...	0

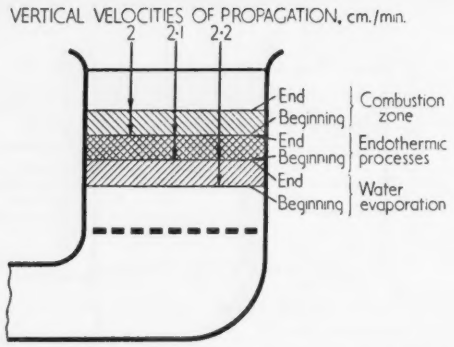

Fig. 2—Speed of propagation of reactions in the sinter bed

and, after the exchange of heat, layer b has a mixed temperature of 750° C. To reheat layer b and its required quantity of air from 750° to 1000° C. the addition of q kilo calories per kilogramme is required:

$$q = mc_1.250° \text{ C.} + Lc_2.250° \text{ C.};$$

where $m = 1$ kg. of cement clinker, $L = 0{\cdot}723$ cu.m. of air, and c_1 and c_2 are the specific heats, $q = 120$ k.cal. per kg. of cement clinker. Layer a at 500° C. exchanges its heat content with the newly entering quantity of fresh air at 0° C.; the mixed temperature is then 250° C. Between layers b and c the above process is repeated; at the same time, layer a cools down further as a result of heat transfer to the fresh air, and so on.

The temperature curve in practice corresponds approximately with the figures given in the last column of Table I.

To attain 1000° C., according to the example, a heat supply of 120 k.cal. $= \frac{1}{2}\ c.1000°$ C. is required for 1 kg. of cement clinker. On the basis of the method of calculation given in Table I, a supply of heat (*e.g.*, by the combustion of coke) of $q = \frac{1}{2}\ ct$ k.cal. would be required to effect the suction after the start of ignition to heat 1 kg. of solid matter to a temperature of t° C. The highest suction temperature would therefore be theoretically

$$t = 2\ q/c,$$

where q is in k.cal. per kg. of ore and c is the mean specific heat.

Since the gases drawn through act primarily as heat carriers and only secondarily as combustion air, it may be concluded that a large excess of air has no great influence on the temperature of the combustion itself. As more gas is drawn through, the heat transfer from above to below becomes faster and the sinter output greater. For this reason down-draught sintering generally takes place under a two- to four-fold excess of air, although in spite of this the melting temperatures of ores and earths can easily be attained, and even exceeded, using only 3–4% of fuel.

With increase of the inner surface of the charge (*i.e.*, finer grain size) the heat-absorption capacity increases, and consequently the efficiency of the suction process, so long as an adequate gas permeability can be maintained during its course.

The down-draught process, therefore, is mainly effected by:

(i) Continuous combustion of a relatively small quantity of fuel within a gas-permeable layer of material

(ii) A temperature propagation within a gas-permeable layer in the direction of the gases flowing through, during which the heat is transferred in part directly by conduction and radiation, but mainly indirectly by the gas flowing from one partial layer to the next. The heat absorption of the raw material and the heat-transfer values have, *inter alia*, a determining influence on the speed of the process.

TRAVELLING SPEED OF REACTIONS IN THE SINTER BED

Characteristic of down-draught suction are the high speeds with which the reactions, such as oxidation, thermal disintegration, and water evaporation, take place. This is due to the rapid direct removal of all gaseous reaction and evaporation products with the waste gases, to the large contact surface between the gas and solid bodies, and to the high combustion temperature, caused by the inherent preheating of the charge and the combustion air. In practice, for each sq. m. of suction surface, 30–110 kg. of coke are burnt per hour (1–2 tonnes/day), 2–5 tonnes of water are evaporated per day, 15 tonnes of $CaCO_3$ are decomposed per day, and 15–40 tonnes of ore are sintered per day.

The evaporation of water and the thermal decompositions (disintegration of lime) are, as is well known, dependent not only on temperature, but also on the partial pressure of the materials to be vaporized and on the effective surface of the charged material. If this partial pressure is constantly kept lower than corresponds to the equilibrium conditions for a certain temperature by rapid and continuous removal of the evaporation products, such as steam and carbon dioxide, the evaporation or decomposition has to continue until all initial material has disappeared. These conditions are especially apparent during suction and, owing to the intake of large quantities of air, the steam escapes from the charge at 50–70° C., and carbon dioxide from the lime (on account of its rapid removal) at about 805° C. (both at a partial pressure of about 200 mm. of Hg), at such a speed

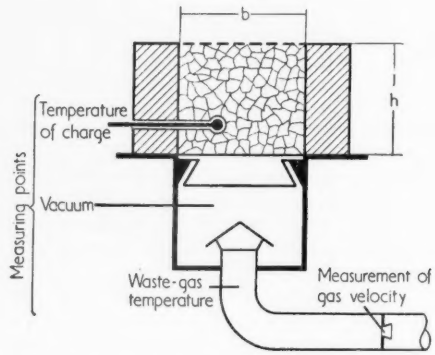

Fig. 3—Apparatus for sinter experiments

as is obtained under other conditions with water only at 100° C. and with lime at 900° C.

The combustion initiated in a down-draught sinter charge by ignition is continued by the intake of air, and for this reason the combustion process and the accompanying reactions, such as evaporation of water and thermal decomposition, continue to proceed in the direction of the gas flow in the charge until they reach the grate. Therefore the speeds at which the distance between the surface of the charge and the grate is covered (*see* Fig. 2) can be distinguished as follows: the vertical speed of propagation of the evaporation of the water, the speeds of ignition, of the endothermic reactions, and of the exothermic main reactions. They must have a certain definite relation with each other. The vertical ignition speed must be as nearly the same as, but never substantially greater than, the vertical roasting or combustion speed, since otherwise a waste-gas stream with an approximately uniform SO_2 or CO_2 content cannot be expected.

When carrying out endothermic reactions, such as the decomposition of lime in a cement raw material, the vertical speed of propagation of the heat-consuming processes within the charge must be higher than that of the exothermic processes (*see* Fig. 2). With the condition reversed, no temperatures higher than the decomposition temperature of the endothermic substance (*e.g.*, 900° C. with lime) can be attained, even with an excessive fuel content of the mixture. This means that the intensive-combustion process cannot develop, the charge in its entire depth gradually becomes glowing, since the ignition continues, and a correct performance of the process becomes impossible.

Endothermic (*i.e.*, heat of decomposition consuming) and also exothermic substances should therefore be present in the charge only in exactly determined quantities, a fact that has to be taken into consideration especially when sintering Portland cement. These proportions may be easily adjusted by the addition of returns (*i.e.*, already treated, partly unsintered, partly sintered, and broken sinter material with a grain size of less than 6 mm.) to the raw material. As a rough guide, no more than half the heat liberated

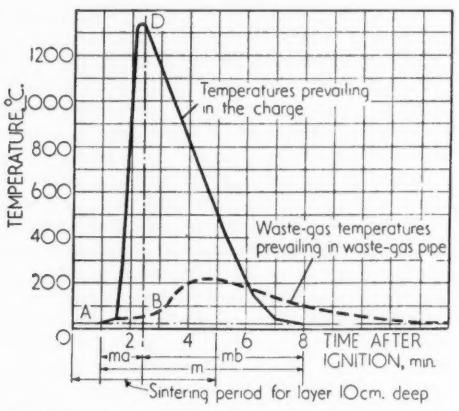

Fig. 4—Curves showing temperatures in the charge and waste-gas temperatures during sintering

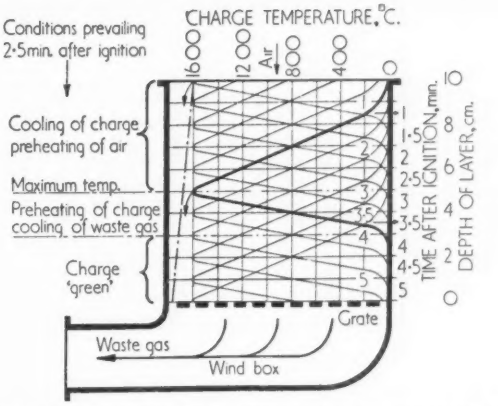

Fig. 5—Temperatures in differential layers during down-draught sintering

in the mixture by the fuel should be utilized for thermal decompositions. In sintering cement, the returns, for example, serve to dilute the lime content of the raw powder, but without changing the chemical composition of the cement. Moreover, the addition of returns not only increases the gas permeability of the charge, but also accelerates the sintering or melting, as during the sintering process the returns have a loosening effect on the surrounding oxides and therefore tend to lower their melting points, thereby helping the formation of the sinter compounds.

HEAT ANALYSIS OF THE SINTERING PROCESS AND THE TEMPERATURE/TIME DIAGRAM

No measurement is more informative for the evaluation of sintering than the temperature within a point of the charge during the process. For series testing of ores the simplest method is to use a sintering pan with a suction area of, say, 1858 sq. cm. One arrangement is shown in Fig. 3, where the thermocouple used for normal tests is attached about 5 cm. above the grate. Besides studying the physical and chemical composition of the mixture, observations are made with the thermocouples of the instantaneous values of the temperature in the charge and in the waste gas, as well as of the vacuum and the quantities of gas extracted. It is characteristic for the temperature of a point within the sinter charge to remain longer at a temperature between 20° and 60° C., even though the combustion is progressing in the sections above the measured point (*see* Fig. 4). As long as the section of the charge whose temperature is being measured still contains water, the temperature cannot rise above 100° C. It is only after the water has evaporated entirely (which generally happens immediately before the start of the combustion) that the temperature rises rapidly to its maximum value and the fuel content of the charge section at the measuring point is burnt. The hot combustion gases at the same time transfer practically their total heat content to the lower layers.

The behaviour of the waste-gas temperature is similar. So long as there is still water in the charge, it will rarely rise above 20–60° C. When all the water has evaporated, the waste-gas temperature rises in

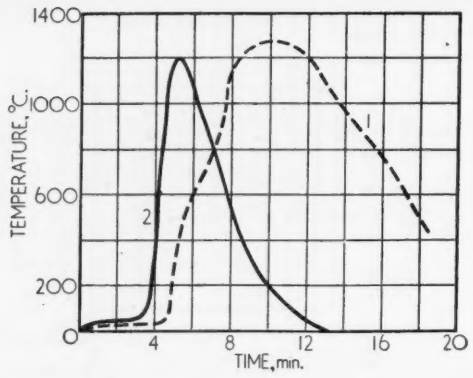

Fig. 6—Temperature/time diagram of an ore mixture (curve 1) without and (curve 2) with 5% lime addition

the same degree as the heat of the lowest layers resting on the grate. The highest waste-gas temperature is measured when the combustion zone reaches the suction grate. At this time the waste-gas temperature would be equal to the sintering temperature if the grate itself did not withdraw part of the heat content of the waste gases.

In Fig. 5 these processes are shown schematically. The unbroken line shows the state of temperatures of a sintering filling $2\frac{1}{2}$ min. after ignition. The surface of the charge has already become cold. Only one section of the charge is heated at a given time, and the highest temperature prevails only in a single horizontal plane. The section adjoining the grate is not heated appreciably at this time; it still contains water, and waste gases saturated with steam, but already cooled down to 20–30° C., flow through it. A portion of the fuel in a charge which is in a state of down-draught sintering is already burnt at a certain point of time of the observation, when another part has not even been ignited. The liberated heat quantities initially serve only to heat one section of the filling and to evaporate the water content; they are later removed with the waste gas, exterior heat losses being disregarded in this case.

The sum of the values of q (heat quantities contained in the heated section), a (heat quantities removed with the waste gas), and w (heat quantities used for the evaporation of the water) is equal to the heat quantities produced by the combustion of a portion of the fuel content b of the charge (inclusive of the ignition heat):

$$b = q + a + w.$$

A short time after the beginning of the down-draught sintering the entire heat produced still remains within the charge; the waste gases escape, for example, at a temperature of 20° C., so that for practical purposes $a = 0$, and $b = q + w$. If the height of the charge layer is assumed to be extraordinarily great, no heat whatever would be removed for the time being, since the hot waste gases would lose their heat continuously to the layers that are still 'green.' The finite height of the layer is the only reason why the heat can be carried away by the waste gas. Instead of $a = 0$, the opposite case may be taken, where $q = 0$. This holds when the air has been drawn through the sintered charge long enough for the latter to be cooled down to fresh-air temperature, so that the air drawn through is no longer heated. The total heat B produced during the sintering has then been removed, together with the total waste-gas content, and moreover the water content of the charge has been evaporated:

$$B = a + w.$$

In this case, only the measurement of the evaporated quantity of water and of the quantity of heat carried away with the total waste gases would be necessary for striking a heat balance.

Under the assumption of a homogeneous course of the down-draught sintering within a certain region of the charge, every point in this region will be subjected to the same rise and fall of temperatures, but, of course, in appropriate sequence. A thermocouple inserted into the charge therefore shows all temperatures according to Figs. 4 and 5 within a certain period of time. This period is proportional to a certain section of the height of the layer. The conception of sintering speed already indicates that layer heights may be converted directly into time, and *vice versa*.

A small grain size of the charged material (*e.g.*, below 2–3 mm.) and of fuel, containing in it 150–260 k.cal. per kg. of ore, give the temperature/time diagram approximately the shape of a triangle; *i.e.*, the maximum appears only for a few seconds (Fig. 4). The period of time of the heating of the charge in this case appears to be shorter than that of the cooling. An explanation for this may be found in the fact that the coherent and partly molten sinter has a smaller surface (and therefore a reduced heat yield to the air) and a better heat conductivity than the as yet unsintered charge consisting of a number of individual grains isolated by gas spaces. A certain fraction of the heat is continuously transferred from the hot sintering region by conduction and radiation to the colder portions of already finished sinter in an opposite direction to the gas stream. A normal course of the sintering process is, of course, a prerequisite.

A practical application of a temperature/time diagram is given by considering the reaction of a silicic-acid-bearing iron ore with an addition of calcined lime. In Fig. 6 the course of temperature during the sintering of the ore without an addition is shown by curve 1. The highest temperature in this case is attained in 10 min. After the addition of 5% of lime, curve 2 shows that the peak temperature was reached after just over 5 min.; *i.e.*, the sintering is almost twice as fast with an addition of lime. The reason for this lies in the fact that the gas permeability of the actual sintering zones, in which the ore is in a half-molten and pasty state, is increased by the addition of lime on account of the shortening of the solidification intervals and changes of the viscosity of the sintering zone.[3–5] Hessle[6] also used the temperature/time diagram as a basis for his work on oxidizing sintering of Swedish magnetite fines.

Practical experience in iron sintering and the literature show that in some cases a mass of coherent partly-molten material is formed in the lower layers

adjoining the grate. To prevent this formation, the lower layers of the filling are provided with less fuel than the upper ones. A charge with a uniformly distributed fuel content will not show the same maximum temperatures in every plane of the horizontal layers, and in some cases the roasting temperature increases or decreases, as is shown by the heat computations mentioned above. An increase in roasting temperature with the progress of the sintering is caused by too high a fuel content; it is due to the increasing and strong preheating of the lower layers by the waste heat of the reaction layers above them. With ores having a low melting point the accumulation of heat in the lower layers often causes an undesired fusing of the ores. In such cases it is of advantage to decrease the fuel content in the lower layers of the charge below the normal. For this reason the so-called ' double-layer sintering ' has been introduced recently in five machines, whereby about 15% saving of fuel has been effected. In cases of lack of fuel in the charge it has been observed that the temperature in the layers adjoining the grate was lowered. This phenomenon appeared in particular where too large a portion of the charge consisted of heat-consuming material, such as lime. In the upper layers this ' excessive ' rise of temperature appeared only under the influence of the hot ignition gases of the ignition furnace.

For a certain fuel content of the filling it is, however, conceivable that the roasting temperature in all strata of the ore layer successively reaches the same height (ideal condition). This is, however, generally not the case, since the fuel (except in cement sintering) is not apportioned in accordance with these factors, but in such a way that much of the sulphur is burnt or that a solid sinter is produced.

The following statements are based on a comparison of the apparent pyrometric heat effect t_s and the maximum temperature t_{max} measured in the sintering charge. The value of the former for 1 kg. of material, derived from the quotient of the effective heat content k and the average specific heat c_m, is $t_s = k/c_m$. The values of t_s in this case generally lie between 250° and 800° C., much lower than the maximum temperatures of 1400–1600° C. To reach the maximum temperature a heat input k_{max} is required which, owing to the utilization of the waste-gas heat, is larger than k by n k.cal. The heat quantity n derives from the fuel that is burnt in the layer arranged above the measuring point and constitutes the amount of heat characteristic for the down-draught sintering process, which shows the extent of utilization of the specific heat.

HEAT DIAGRAM

Figure 7 shows in simplified form the heat balance for a certain case of burning Portland cement; it will be used as a computation example, since the specific heat of cement clinkers is well known. The mixture consisted of 1·52 parts by weight of raw meal corresponding to 1 part by weight of clinker, 2·88 parts by weight of returns, 0·18 parts by weight of coke fines (corresponding to 1200 k.cal., inclusive of ignition), and 0·48 parts by weight of water (corresponding to about 9·5% of water in the mixture).

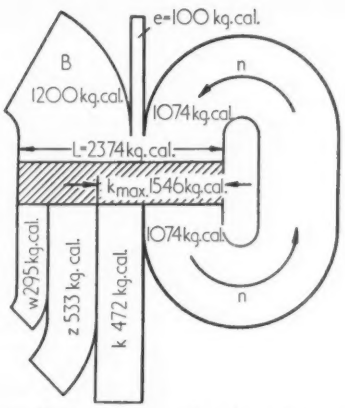

Fig. 7—Heat diagram of a sintering process

The actual fuel consumption ($B = 1200$ k.cal. per kg.) corresponds to a value that has been determined experimentally by the author for raw cement powder rich in lime. At the formation of the clinker compounds, at least 100 k.cal. per kg. of clinkers, of exothermic reaction heat e, are liberated.

According to this example, evaporation of the 0·48 kg. of water per kg. of clinker requires (at about 60° C.)

$$w = 0{\cdot}48 \times 615 = 295 \text{ k.cal. per kg.}$$

To drive the carbon dioxide out of the raw meal about 509 k.cal. per kg. of clinkers are required, and to drive the water of crystallization out of the clay content of the raw meal about 24 k.cal. per kg. are needed, so that for thermal decomposition, $z = 533$ k.cal. per kg. of clinkers.

The heat requirement k_{max} for heating the dry charge to 1400° C. consists of

(i) Heating 1·52 kg. of raw meal to 900° C.: 341 k.cal. per kg. of clinkers
(ii) Heating 1·00 kg. of clinkers resulting from (i) from 900° C. to 1400° C.: 155 k.cal. per kg. of clinkers
(iii) Heating 2·88 kg. of returns to 1400° C.: 1050 k.cal. per kg. of clinkers.

Altogether $k_m = 1546$ k.cal. per kg. of clinkers.

The theoretically required total heat $L = k_{max} + w + z = 2374$ k.cal. per kg. The effective waste heat $n = L - (B + e) = 1074$ k.cal. per kg. The heat losses by waste gas, clinkers, conduction, and radiation $k = k_{max} - n$ or $(B + e) - (w + z) = 472$ k.cal. per kg.

For each kg. of cement clinkers, about 6–10 cu.m. of suction air is required which, heated to 70–100° C. above the initial temperature, escapes as waste gas, in this case unused. Heating 9 cu.m. of air by 100° C. requires $a = 9 \times 0{\cdot}31 \times 100 = 279$ k.cal. per kg. of clinkers, which are covered by $k = 472$ k.cal. per kg. The waste gases (temperature about 100° C.) also contain about 0·51 cu.m. of carbonic acid, derived from the raw meal and the combustion of coke, with which about $c = 21$ k.cal. per kg. are discharged.

For the clinker waste heat and losses by conduction and radiation there remain $k - a - c = 172$ k.cal. per kg. of clinkers. Of these about 112 k.cal. may be clinker waste heat (average temperature of 1 kg.

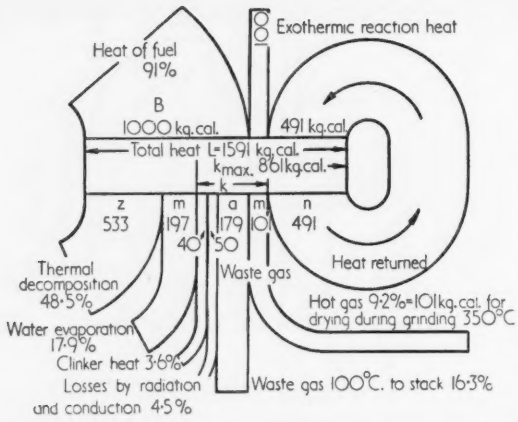

Fig. 8—Heat diagram for 1 kg. of clinker (Lurgi cement-sintering process)

of clinkers and 2·88 kg. of returns to 150° C., of which about four-fifths of the sinter material may occur practically cold and one-fifth glowing) and 60 k.cal. per kg. of clinkers conduction and radiation losses.

The down-draught sintering intensity $i = k_m/k$ ($= 1546/472 = 3{\cdot}27$ for the above example) for certain cases may reach values of up to 5 and higher. Figure 8 shows a heat balance taken from Lurgi cement-sintering practice.

These statements show that the high heat effect of down-draught sintering is caused by the self-preheating of the solid and gaseous combustion constituents according to the process. The resulting and possible temperatures are dealt with in the next section.

THEORETICAL COMBUSTION TEMPERATURES IN ORE SINTERING

In roasting sulphidic ores in mechanical furnaces, owing to the thermodynamically good functioning of the mechanical open-hearth or roasting furnace, the temperature is limited not only by the heat losses but also by the heat resistance of the refractory iron material and the sintering point of the ores. These limitations are partly obviated by down-draught sintering, since in this case the ore can be heated up to and above the melting temperature for fractions of seconds. For example, in the sintering of ores temperatures of up to 1660° C., and of refractory materials up to 2000° C., have been observed.

To calculate the temperatures that may be reached in sintering, a very simple practical formula has been used, according to which the temperature is given as

$$t = \frac{2q}{c} \text{ °C.}$$

where q is in k.cal. per kg. of ore, and c is the mean specific heat for t °C. of the material to be heated. According to this equation, the following materials may be heated to 1000° C.:

1 kg. of Fe_3O_4, with about 130 k.cal. in the form of admixed fuel
1 kg. of cement clinkers, with about 120 k.cal. of admixed fuel
1 kg. of ZnO, with about 74 k.cal. of admixed fuel.

Corresponding values have already been attained in practice.

With down-draught sintering processes the reaction temperature may theoretically rise to a height at which a combustion product begins to dissociate. In practice this temperature is not generally reached, since a limit is set for the combustion temperature by the heat losses arising from the waste-gas losses, by conduction and radiation, by evaporation work to be done on volatile constituents of the charge (provided that the down draught is not interfered with by the formation of lumpy agglomerations and the throttling of air caused by them). In the computations the melting heat occurring at the appearance of the liquid phase is not considered, since it is utilizable again for the lower layers, because the heat quantities liberated at the solidification have to be transferred to the fresh air. The same applies to some extent for the heat of vaporization of oxides, in so far as the vaporization products may be condensed again in the lower layers. In this case, the heat of condensation contributes to the preheating of these layers.

Nevertheless, the requisite heat and time for the vaporization of oxides and also for the melting causes a considerable restraint on any further rise in temperature, not to mention the deterioration of the gas permeability, caused by the appearance of the liquid phase within the solid constituents of the charge.

With the same fuel content and the same physical condition of two different charges, the charge with the lower specific heat will naturally produce a higher temperature. The limit of the fuel content downward is dependent on one hand on the specific heat and the heat conductivity of the charge, the ignition temperature, and the combustion speed of the fuel (which slows down with decreasing temperature), and on the other hand on the speed of the heat transfer within the filling. The more time that is required for the combustion of a certain quantity of fuel, the more effective the exterior heat losses by conduction and radiation become; the heat therefore has a better chance to be distributed over the entire charge, instead of being restricted to one region. If the heat losses

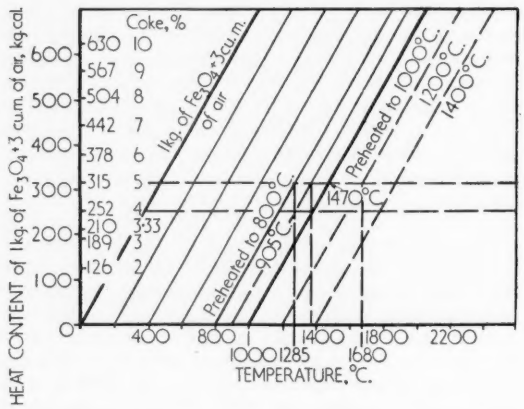

Fig. 9—Dependence of sinter temperature on degree of heating and quantity of fuel

in a given time become just as large as the heat quantities produced, the increase in temperature must come to a standstill. The faster the combustion, the less influence the exterior heat losses will have on the course of the process.

The ratio $t = 2q/C$, valid for an ideal charge, is also applicable for approximate temperature calculations. If, therefore, a theoretical temperature of more than 2000° C. is calculated for an ore, the charge will receive too much, too coarse, or unsuitable fuel, or the gas permeability or the surface of the ore will be insufficient.

When the heat content of the initial material is plotted against temperature, the theoretical combustion temperature can easily be determined graphically, since the heats of reaction are known. Figure 9 shows such a diagram; the mean specific heat of the initial material is plotted as a linear function of the temperature as it appears in the first approximation.

The quantity of heat developed by the reaction

$$2Fe_2O_4 + \tfrac{1}{2}O_2 \rightleftharpoons 3Fe_2O_3 \pm 61{\cdot}7 \text{ k.cal.}$$

has been neglected in this case. The diagram merely shows the connection between attained sintering temperature and the degree of self-preheating (dependent on the capacity of heat absorption of the sintering bed).

Figure 9 shows graphically the combustion of coke in ore. On the ordinate are plotted the heat content for 1 kg. of Fe_3O_4 + 3 cu.m. of air and the corresponding percentage content of coke fines (coke with 6309 k.cal.). The computation is based on the figure of 3 cu.m. of combustion air, found in practice to be necessary for 1 kg. of sinter (iron-ore sintering). Theoretically, with 5% of coke only 475° C. combustion temperature may be obtained with no preheat; 1680° C. is only possible with a preheating of air and ore to 1200° C., which in down-draught sintering is practicable. With 8% of coke a combustion temperature of 1800° C. would be attained if the air and the ore were preheated to about 1135° C. (by means of waste-heat utilization).

Figure 9 can also be used to determine the fuel consumption theoretically required for attaining a specific temperature, while taking in consideration a certain preheating of the initial material. If, for example, 1 kg. of ore + 3 cu.m. of air are to be heated to 1000° C. assuming a preheat of the initial material to 800° C., Fig. 9 shows that about 130 k.cal. are needed for this purpose, which are contained in about 2·07% (G.T.) of coke with a heating value of 6300 k.cal. to which 100 parts of ore are added. Instead of 2·07% of coke, the 4·22% of sulphur contained in pyrite may also be used. Should, for example, a preheat of only 400° C. be attained, about 400 k.cal., corresponding to 6·3% of coke fines, would be required to attain a sintering temperature of 1000° C.

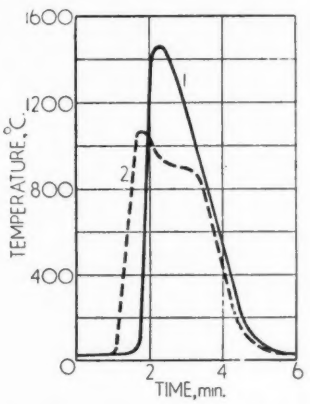

Fig. 10—Temperature/time diagram of an ore mixture with coke of grain size (curve 1) 0–1 mm. and (curve 2) 3–6 mm.

SIGNIFICANCE OF FUEL GRAIN SIZE

The influence of the fuel grain size on the sintering temperature is illustrated by Fig. 10, taken with a 3·3% addition of fuel. In both cases an ore with a screen size of 2–6 mm. has been used; in test 1 coke fines with a grain size of 0–1 mm. were used and in test 2 coke fines with a grain size of 3–6 mm. With the finer coke a maximum temperature of over 1400° C. was attained, and with the coarse coke a temperature of 1050° C. The intensity of the sintering in test 1 amounted to 2·0, and in test 2 to only 1·36. The diagram shows clearly that the coarser coke requires a longer combustion time. The fine coke is distributed much more uniformly in the charge, and the size of its surface is a multiple of that of the coarser coke. It is for this reason, and furthermore because abundant combustion air is available, that it burns intensely in such a very short time. Coarse-grained fuel requires a combustion time proportional to its grain size, as curve 2 of Fig. 10 clearly shows.

After the maximum temperature has been reached, utilizing the preheated air, a rapid decrease in temperature is observed at first, followed by a retardation of the drop in temperature, because of the afterburning of the fuel as yet unburnt; during this time the latter receives much less preheated air, since the sintering layer above it has cooled down in the meantime. The combustion time of the 3–6-mm. coke is about 2 min. according to the diagram. In these 2 min., however, the combustion process has already progressed about 5 cm. further, and the desired concentration of the heat on a layer as thin as possible is impeded. Coke that is too coarse decreases the intensity of the combustion considerably, and leads to afterburning on discharging and furthermore to the formation of unwanted 2CO out of CO_2 + C, since the burnt gases have still to flow continuously over the still unburnt glowing pieces of coke.

It is obvious therefore that fuel with a grain size below 3 mm. must be used for sintering, because this not only shortens the time of sintering (*i.e.*, increases the output), but also saves considerable fuel. Sinter plants which have more than 0·5% of CO in the waste gases are certainly operated with too coarse a coke.

EXCESS OF AIR

In practice, down-draught sintering requires about 3 cu.m. of air per kg. of sinter. The excess air of the combustion process can be expressed roughly as two-

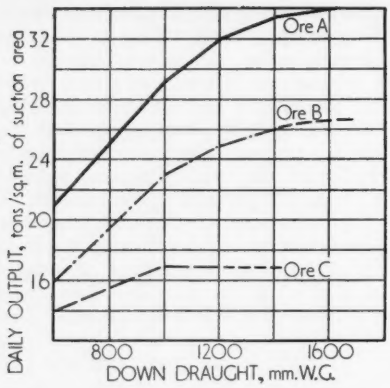

Fig. 11—Dependence of sinter capacity on degree of vacuum

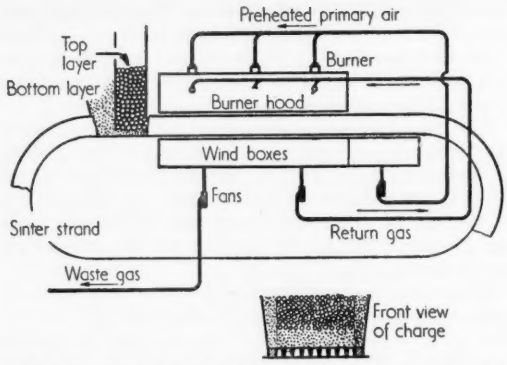

Fig. 12—Gas-heated sinter band (Lurgi system)

to six-fold, and it must be taken into consideration that the sintering waste gases often also contain cooling air which, in the last suction boxes, is drawn through the sintering bed without effecting combustion. The CO_2 content of the waste gases therefore generally lies in practice between 4 and 8%.

A certain excess of air, rather high in comparison with other combustion processes, has no unfavourable effect on down-draught sintering, and actually improves the combustion. An explanation for this lies in the fact that the combustion gases transfer practically all the heat absorbed by them to the lower layers of the charge while passing through them, so that they act solely as heat carriers from above downwards as long as the sintering zone has not reached the grate. That the gases actually are heated up to the reaction temperature, although they stay only momentarily in the charge, is shown by the fact that the waste-gas temperatures rise very high just before the end of the sintering process.

If the supply of air is increased (*e.g.*, by increasing the vacuum), correspondingly more air per second flows through the charge as before, and the heat transfer by the gases downward is therefore accelerated. Moreover, the intensity of the combustion is increased as a result of the greater excess of oxygen and the accelerated discharge of combustion products. The consequences are increased vertical sintering speed v and therefore higher output.

The limits of the gas speed, and also of the excess of air, are only reached either when the time the gases stay in the charge is not sufficient to allow as complete as possible a transfer of the heat content of the gases to the still ' green ' lower layers, or when the surface of the charge per volume is too small (too coarse-grained) to be able to absorb the heat content of the gases completely in the short time available.

In most cases, however, the viscidity of the sintering zone when in the melting condition limits the transit of the air. As shown in Fig. 11, the output of several ores has been increased by increasing the vacuum; practical experience, however, has shown that an increase above 1000 mm. water column offers economical advantages only with very few types of ore.

This survey of the thermal and physical processes occurring during down-draught sintering makes no claims to completeness. For instance, a more detailed description of the conditions in relation to gas permeability has been omitted, since adequate information in this respect may be found in the literature. Moreover, the use of solid fuel mixed with the ore is not essential; the latter may be replaced partly (mixed firing), or even totally (gas heating), by gaseous fuel, such as blast-furnace or town gas, by oil, or by coal-dust firing. A sintering plant in the Saar district has been changed to a mixed-fired sintering process proposed by Lurgi; about half the coke fines have been replaced by blast-furnace gas, which is burnt in a large firing hood over the first part of the sintering machine. The degree of oxidation of this sinter is appreciably higher than that produced solely by the addition of coke.

It is of interest in regard to gas permeability that a process of pellet hardening on gas-fired sintering bands has been developed in the U.S.A. and independently in Germany by Lurgi.

During roasting of the pellets, the physical laws governing the flow of gas through ball-shaped heaped materials are completely obeyed and the process is therefore subject to accurate computation, since the two incalculable factors of the changes in gas permeability by grain decomposition at the drying-out or by the formation of a fusing sintering zone are obviated.

This paper will have served its purpose if with its help the practical engineer is able to recognize his problems in connection with the sintering of ores more clearly, and can thereby solve them in such a way that the efficiency of the plant is improved.

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The Mechanism of Reduction of Iron Oxides

By J. O. Edström

SYNOPSIS

The reduction of pure, natural hematite and magnetite crystals by hydrogen and carbon monoxide was studied mainly with the aid of the microscope. The more rapid reduction of Fe_2O_3 is explained by an earlier and more extensive formation of pores in the reaction products. The low strength of pure hematite ores and of well-oxidized, burnt, rich-ore pellets during reduction with carbon monoxide is due to the formation of pores, accompanied by a considerable expansion and a disintegration of the structure.

The minimum reaction velocity above 600° C. when Fe_2O_3 and Fe_3O_4 are reduced by hydrogen is associated with the appearance of dense continuous layers of wüstite.

The mechanism of reduction and the causes of the different pore-formation tendencies are discussed. 833

THE equilibrium conditions in the reduction of the oxides of iron by CO or H_2, and the reduction velocities of certain types of ore at different temperatures, are well known through the work of several investigators. Hematite ores are generally reduced with a higher velocity than magnetite ores, and increasing the degree of oxidation of the ore by roasting, burning of briquettes, or sintering, improves its reducibility, in spite of the increased amount of oxygen removed during reduction.

The change of volume of iron ores during reduction has also been investigated. It is well known that different ores show big differences in this respect, and also that there is a relationship between volume change, reducibility, and strength during reduction. The strength is important, especially in connection with production of sponge iron in a shaft furnace by the Wiberg method.

The work of Wiberg,[1, 2] on the microstructure of a partially reduced magnetite crystal, was continued by the author, in collaboration with S. E. Eriksson, at the Royal Institute of Technology, Stockholm, under the supervision of Professor Wiberg.[3, 4] The work comprised a study of the microstructure of magnetite and hematite crystals partially reduced by CO. The author has now studied the reduction mechanism microscopically as well as by X-ray crystallographic methods. The relation between microstructure and reduction velocity was also studied, and an explanation of the difference in reducibility of magnetite and hematite, and the different volume changes during reduction, is given.

PREVIOUS INVESTIGATIONS

The reduction of iron ores has been studied by many investigators. Their work has been partly practical, dealing with reduction processes, and partly theoretical, often involving comprehensive laboratory experiments. Of the previous practical investigations, mention may be made of the work of Bohm,[5, 6] Wiberg,[7, 8] Tigerschiöld,[9] Améen,[10] Cavanagh,[11] Eketorp,[12] and Stålhed.[13] An account of the main Swedish work before 1940 has been given by Wiberg,[1] and a very comprehensive bibliography by Specht and Zapffe[14] (1946). The theoretical investigations are discussed in the following sections.

Equilibrium Diagrams

Iron has three oxides, hematite (Fe_2O_3), magnetite (Fe_3O_4), and wüstite (Fe_xO, where x is a little lower than 1). Magnetite and wüstite exist within certain concentration ranges. Some details of the equilibrium diagrams are still controversial,[15, 16] especially the wüstite range. Figure 1 shows the diagram according to Darken and Gurry,[15] which is generally accepted in American literature.

The oxide phases in equilibrium with different gas mixtures of CO and CO_2 and of H_2 and H_2O are shown in Fig. 2, which gives the equilibrium diagrams Fe–O–C and Fe–O–H, mainly as compiled by Wiberg.[1] The curves for the oxygen content of wüstite are according to Sambongi[17] and Takeuchi and Igaki.[18]

Crystal Structure of Iron Oxides

Wüstite has a cubic lattice of NaCl type, *i.e.*, close-packed oxygen lattice with iron ions arranged in the octahedral interstices between the larger O^{2-} ions. The ion radii are, according to Pauling, $Fe^{2+} = 0{\cdot}75$ Å., $Fe^{3+} = 0{\cdot}60$ Å., and $O^{2-} = 1{\cdot}40$ Å. Wüstite is unstable below 570° C., when it decomposes eutectoidally into α-Fe and Fe_3O_4. However, it is easy to undercool the wüstite and prevent the decomposition $4FeO \rightarrow Fe_3O_4 + Fe$, but pre-eutectoid Fe_3O_4 precipitates fairly quickly. The range of existence of wüstite is fairly large but does not include FeO; the oxide always has a higher oxygen content, as the iron lattice contains 5–11% vacancies.[15] Electric neutrality is obtained by the formation of trivalent iron ions equal to the double number of vacancies. According to Jette and Foote,[19] the lattice parameter varies linearly with oxygen content, from 4·3010 to 4·2816 Å., corresponding to contents of 22·27 and 23·92%, respectively. The corresponding observed densities were 5·728 and 5·613 g./c.c.

Magnetite (Fe_3O_4) also has a cubic lattice. This is of the spinel type, with $a = 8{\cdot}38$ Å.[20] It has a close-packed cubic lattice of O^{2-} ions with the smaller Fe^{2+} and Fe^{3+} ions distributed in the interstices. Of these, 8 Fe^{3+} are in tetrahedral and 8 Fe^{3+} + 8 Fe^{2+} are in octahedral positions. The unit cell contains 32 oxygen ions and 24 iron ions and has 64 tetrahedral and 32 octahedral interstices. This oxide

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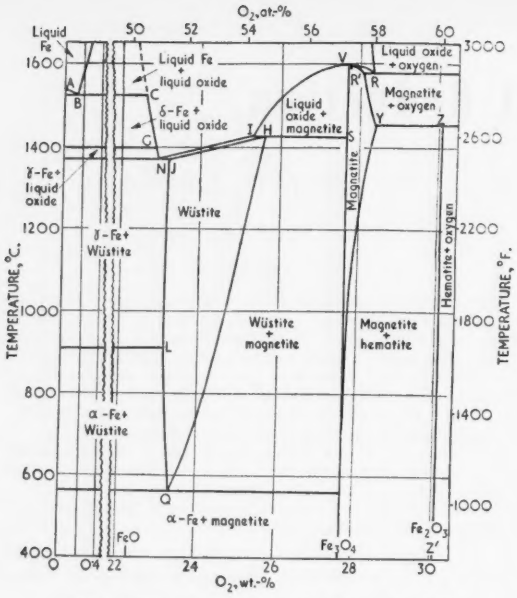


Point	Temp., ° C.	O_2, %	p_{CO_2}/p_{CO}	p_{O_2}, atm.
A	1539			
B	1528	0·16	0·209	
C	1528	22·60	0·209	
G	1400*	22·84	0·263	
H	1424	25·60	16·2	
I	1424	25·31	16·2	
J	1371	23·16	0·282	
L	911*	23·10	0·447	
N	1371	22·91	0·282	
Q	560	23·26	1·05	
R	1583	28·30		1
R′	1583	28·07		1
S	1424	27·64	16·2	
V	1597	27·64		0·0575
Y	1457	28·36		1
Z	1457	30·04		1
Z′		30·06		

* Values for pure iron

Fig. 1—The iron–oxygen system

also shows a gradually decreasing lattice parameter with increasing oxygen, indicating the presence of vacant positive lattice points,[20] probably in octahedral positions. The homogeneity range is smaller than for wüstite, but is definitely established.

The magnetite lattice is very similar to that of wüstite; the oxygen lattices correspond but the arrangement and number of iron ions is different. Goldschmidt[21] calculated the lattice dimensions of a pseudo unit cell which would include both wüstite and magnetite, and found that the linear decrease of lattice parameter with increased oxygen, which Jette and Foote had found within the homogeneity range of wüstite, could be extended to include also a composition corresponding to magnetite with its observed lattice parameter. Buinov, Komer, Zhuraleva, and Chufarov[22] found that the reduction of Fe_3O_4 to FeO is crystallographically reversible, *i.e.*, the single-crystal structure is not destroyed by repeated oxidations and reductions. The crystalline transformation proceeds along parallel axes.

Hematite (Fe_2O_3) may exist in two modifications with different lattices: (1) α-Fe_2O_3 is of rhombohedral corundum type ($a = 5{\cdot}42$ Å. and $x = 55° 14'$), the O^{2-} ions are arranged in a close-packed hexagonal lattice and two-thirds of the octahedral interstices are occupied by Fe^{3+} ions. The oxide has a small oxygen deficit, probably because of O^{2-} vacancies, but possibly also due to iron ions in additional interstitial positions. (2) γ-Fe_2O_3 is, like magnetite, a cubic spinel type, with $a = 8{\cdot}32$ Å., and has an average of $21\frac{1}{3}$ Fe^{3+} and 32 O^{2-} per unit cell.[23] It may be obtained by dehydration of γ-FeOOH below 500° C. or by oxidation of Fe_3O_4 below 400° C. Transformation of α-Fe_2O_3 to γ-Fe_2O_3 has not been observed and it appears that γ-Fe_2O_3 is a metastable phase formed only during dehydration or oxidation.

The orientation relationships between Fe_3O_4 reduced on Fe_2O_3 and the latter do not seem to have been determined. A transformation of α-Fe_2O_3 with hexagonal lattice to cubic Fe_3O_4 should, however, in contrast to the transformation of Fe_3O_4 to Fe_xO, give rise to a lattice distortion which might lead to a breaking up of the structure.

The lattice orientations in connection with oxidation of magnetite to hematite are well known. Gruner showed that the (0001) oxygen ion plane in α-Fe_2O_3 coincides with the (111) plane in Fe_3O_4 and that the directions [10$\bar{1}$0] in the former and [110] in the latter are perpendicular to each other.[24] When wüstite is formed on α-iron, the (001) plane for the iron ions is common to both phases and the [010] direction in wüstite is parallel to the [110] direction in α-Fe.[25, 26] The phase relations between γ-Fe and wüstite do not seem to have been studied.

At room temperature the specific volumes for the different phases are approx. : 0·272 c.c. Fe_2O_3, 0·270 c.c. Fe_3O_4, 0·231 c.c. FeO (23·5% oxygen), and

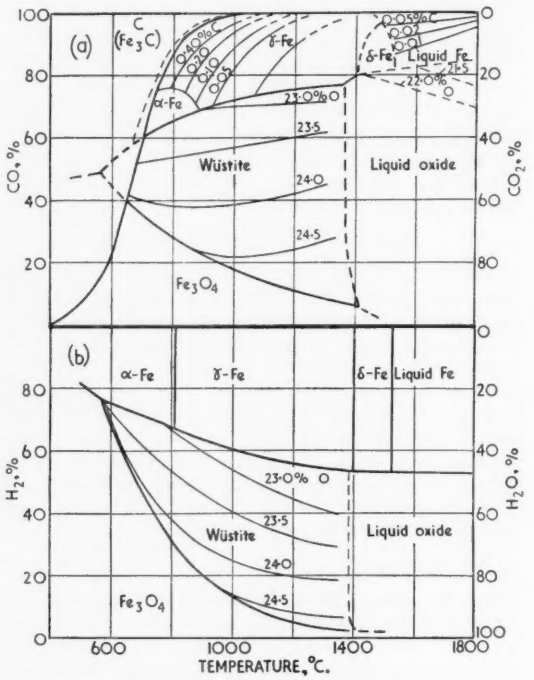


Fig. 2—(*a*) Iron–oxygen–carbon system at 1 atm.; (*b*) iron–oxygen–hydrogen system

0·128 c.c. Fe, per gram of Fe. The conditions for formation of continuous layers are thus fulfilled, *i.e.*, the outer component has a higher specific volume than the inner one.[27] When the oxides are reduced, porosities should appear if the external volume does not decrease, but on reducing Fe_2O_3 to Fe_3O_4 the porosity should be very insignificant.

Reduction Mechanism

The reduction of iron oxides has often been treated as a gas diffusion problem, especially diffusion through the reduced metal (sponge iron).[28, 29] The preparation of specimens was often too difficult for studying microscopically the solid phases during reduction. With few exceptions, the true structures were not obtained, and it was not possible to distinguish between reduction porosity and grinding faults. Materials with high original porosity were often chosen, which was also misleading.

It has long been known that reduction of magnetite at higher temperatures proceeds by way of the phase wüstite to metallic iron and penetrates towards the centre of the ore sample so that concentric layers of the different phases appear. At least from a macroscopic point of view, the reaction is a topochemical one.[28, 30, 31]

The phases do not always appear in the expected order. Gellner and Richardson[32] found that, at 900° C. with H_2 as reducing agent, wüstite prepared by oxidation of pure iron is, after formation of a thin surface film of metal, reduced from inside towards the surface, where the base metal meets the oxide layer and where the oxide probably is least stable.

As already mentioned, it is to be expected that the reduction products from the iron oxides should be porous. However, Wiberg[1, 2] studied the reduction of a natural magnetite crystal with CO at 1000°C. and could not observe any porosity in the wüstite layer, in spite of the differences in specific volume, and he assumed that the mass transport across this layer occurred in the solid phase. Bitsianes's micrographs[31] of reduction products from Fe_2O_3 and Fe_3O_4 do not give any definite information on the porosity of the different phases, as the specimens already contained considerable porosity or cracks.

Phenomena in the solid phase have thus become more and more observed in connection with reduction processes, which may now largely be interpreted as reactions in the solid phase. Besides the interchange of gas between the oxide/metal reaction surface and the stream of reducing gas, there are the elementary processes of diffusion in the solid phases and reactions in the phase boundaries. The influence of the phase-boundary reactions was pointed out by Stålhane and Malmberg,[30] who found that the reduction of Fe_3O_4 to wüstite was considerably faster than from wüstite to Fe, the latter reaction determining the total speed of the reduction.

The mechanism of oxidation has been studied more than that of reduction, and it was shown long ago that the majority of oxidation processes follow the so-called parabolic law,[2, 27] *i.e.*, the oxide layers are dense and their thickness is proportional to the square of the reaction time. This means that diffusion through the oxide layer is the only rate-determining process if the concentrations or chemical activities at the phase boundaries are independent of time. A possible assumption is that oxygen diffuses from the outer surfaces across the oxide layer, and that oxide is then formed in the surface between oxide and metal. But it is also possible that the metal migrates across the oxide layer to react with oxygen on the outer surface. Wagner[33] has mathematically calculated the velocities of reactions between metals and oxides, and sulphides and halogens, and has experimentally verified his results. The mass transport consists of a migration of ions and electrons, while transport of electrically neutral atoms or molecules may be neglected. Two extreme cases may appear as follows:

Migration Processes in an Oxide Layer (*Wagner*[33])

Metal | Oxide | Oxygen

A { Metal ions → ; Electrons → }

B { Oxygen ions ← ; Electrons ← }

The transport of ions and electrons may proceed according to the mechanisms proposed for diffusion, *e.g.*, *via* vacancies or interstitially.

The mechanism of transportation in the oxidation of iron appears to be fairly well known. Microscopical measurements[34] and radioactive tracers[35, 36] have shown that the transportation through the wüstite layer is a diffusion of iron ions. Opinions differ on the transport across the magnetite and hematite layers, but according to the latest publication[36] it should in the former case proceed by means of iron ions and in the latter by oxygen ions, as might be expected from the vacancies present.

Wagner also studied the reduction of wüstite.[37] Gellner and Richardson[32] explained their observation that wüstite obtained by oxidation of iron at 900° C. may be reduced from the centre outwards by the following assumptions, which agree with Wagner: (1) Removal of O_2 by H_2 from the outer surface of the oxide, causing supersaturation of the oxide with iron; (2) nucleation of this excess iron, causing thin layers of metal to be formed on the surface; (3) retardation of the further reduction caused by the surface metal, which prevents contact between H_2 and the oxide, and diffusion of water vapour formed; (4) diffusion of excess iron formed in the outer layers of the oxide during the continuing reduction into the body of the oxide; (5) (if the temperature is high and diffusion sufficiently rapid) preferential growth of iron outwards from the base metal where the oxide may be expected to be least stable and the disposition for nucleus formation the highest. When a completely oxidized specimen was heated, an equalization of the composition of the oxide and a recrystallization was obtained; the oxide grains were reduced uniformly over the whole surface, and the material in the centre and near the surface of the specimen behaved in the same way.

Influence of Different Factors on Rate of Reduction

A considerable lowering of the reduction rate is

obtained at temperatures of about, or a little above, 600° C., both with H_2[14, 38, 39, 40] and with CO[38] as reducing agents. Ramseyer[41] and Wienert[38] believed that the appearance of wüstite as an intermediate product retarded the reaction. Others, in later publications, have shown that the retardation, which first appears above the eutectoid point of wüstite, may be caused by a decreasing diffusion velocity for water vapour through the iron phase,[40] which recrystallizes rapidly above 600° C.[42, 43] Also, slightly above 900° C., a discontinuity in the increase of the reaction rate with temperature has been observed.[30, 40, 44] Tikkanen[40] explained this as a result of a lower diffusion velocity for H_2O in γ-Fe than in α-Fe. Stålhane and Malmberg[30] also pointed to the α–γ transformation. They found that if CO is added to the reduction gas, the discontinuity temperature is lowered in parallel with the lower boundary of the stability range of austenite caused by the carburization.

Reduction by H_2 is, at least at the beginning of the reaction, faster than by CO, as would be expected from the diffusibilities of the gases. Tenenbaum and Joseph[45] state that a 14-mm. cube of dehydrated limonite ore (a very porous specimen) was reduced four times faster with H_2 than with CO at 1000° C. Similar times were obtained by Stålhane and Malmberg[30] when reducing 1-mm. ore grains. When reducing a Kiruna magnetite at 1000° C., Wiberg[1, 2] found that reduction was faster by H_2 until about 75% of the ore's oxygen had been removed. Then the H_2 reduction rate decreased considerably more than that of CO, and the time for complete reduction was longer with H_2. This was explained as follows: CO carburizes the iron reduced, which, in the shape of thin shells, surrounds the small wüstite grains. Reaction between carbon in the austenite and oxygen in the wüstite produces a high gas pressure at the boundary surface, and the iron shell is broken. Thus, gas exchange at the oxide/metal reaction surface is facilitated. With H_2, a similar effect may appear, as H_2, but not water vapour, may diffuse through the iron shell. The pressure at the boundary surface is then lower and the bursting power is therefore not so pronounced.

Even a small CO_2 content in the gas very much retards the reduction of wüstite to iron; H_2O also has a strong retarding action, but less than CO_2. The reduction of magnetite to wüstite has, on the other hand, been shown to proceed at the same velocity with additions of 0–30% of CO_2 to the reducing gas.[30] The influence of other gases has also been studied.[46]

It has been found that a relatively low critical gas velocity exists, above which a further rise has very little influence. Udy and Lorig[39] found this velocity to be a function of the temperature and of the bed thickness and grain size of the ore, but not of the degree of reduction. The influence of increased gas pressure has been studied.[45, 47, 48, 49] According to Marek, Bogrov, and King,[49] the total oxygen removed, and the momentary composition of the gas that leaves the bed, are both constant if a given weight of CO passes a hematite ore specimen, irrespective of velocity and pressure of the gas. Thus, the mass rate of flow of the gas is the deciding factor.

The chemical composition of the raw material has great influence on the reduction. As already mentioned, in blast-furnace practice an oxidized magnetite ore is more quickly reduced than an unoxidized ore, and the reducibility of sinter is increased with the degree of oxidation. These results have been controlled by laboratory experiments.[6, 29, 50, 51] Even a small change in the degree of oxidation influences the reducibility considerably, and experiments therefore require great care in the choice of material. The degree of oxidation is defined as the ratio between the oxygen content bound to iron present in the specimen and that which should be present if all iron were bound as Fe_2O_3.

No detailed investigation on the causes of the differences in reducibility between Fe_2O_3 and Fe_3O_4 appears to have been made. Hofman,[44] observing a considerable difference in the reducibilities, questioned whether Fe_3O_4 is formed as an intermediate stage at the reduction of Fe_2O_3 with pure H_2. Other workers are of the opinion that Fe_3O_4 as an intermediate stage would have a certain porosity, and that the reduction of this magnetite with greater surface per unit volume should therefore proceed faster than that of a dense magnetite.[29]

Concerning the influence of oxidation in blast-furnace reduction, Wiberg[8] pointed out that the reduction of Fe_2O_3 to Fe_3O_4 is exothermic, while the reduction of Fe_3O_4 to wüstite is endothermic. If the ore is well oxidized to Fe_2O_3, the initial reduction will therefore hasten the heating of the ore, while when the ore consists of magnetite the heating is retarded.

Oxides other than those of iron may influence reducibility. Mixtures of basic oxides may, in certain cases, increase reducibility.[44] Oxides of an acid character lead to the formation of slag at temperatures below 600° C.,[52] and the iron compounds formed may hardly be further reduced by gas.

The reduction of ore containing carbonate and hydrate is retarded at temperatures below the decomposition temperature of the compounds. The slow generation of CO_2 or H_2O prevents the supply of fresh gas to the oxide/metal reaction surface, and high partial pressures of CO_2 and H_2O are therefore obtained at this surface. Such ores should be roasted or burned before reduction.

The external shape of an ore with a given chemical composition influences its reducibility considerably. Joseph[53] found that, with all other factors constant, the time for complete reduction of a specimen was inversely proportional to the porosity of the untreated sample. The porosity and the cracks formed during the reduction are also important. An ore with high original porosity may be reduced more slowly than a dense ore if the latter has a higher tendency to cracking and formation of pores during heating to reduction temperature and during the reduction.[54] The influence of the particle size in reducing magnetite concentrates by H_2 at 600° C. was investigated by Specht and Zapffe,[14] who summed up their results in the formula:

$$\text{Log } D = 0{\cdot}0375\ M - 3{\cdot}04,$$

where M is the time in minutes for complete reduction, and D is the particle size expressed in mesh diameter. In reduction experiments with H_2 at 800° C., Lewis[55]

found that the time for reduction of carefully ground cubes of a certain ore is directly proportional to the volume of the cubes.

PRESENT INVESTIGATION

Composition and Specific Gravity of Materials

Natural single crystals of Fe_2O_3 and Fe_3O_4 were used, being suitably dense for structural studies. Most of the crystals were from Malmberget and were put at the author's disposal by the Department of Mineralogy and Geology of the Royal Institute of Technology, Stockholm.

It is difficult to choose suitable crystals, especially of Fe_3O_4, since they are often partly oxidized and contain (in spite of ideal external form) small amounts of Fe_2O_3, which considerably influences the reducibility. Sometimes inclusions occur in apparently ideal crystals.

Several different types of crystals were tested; the compositions most generally used were as follows:

	Fe_2O_3 II	Fe_3O_4 I	Fe_3O_4 VII
Fe_{tot}, %	69·89	72·42	72·24
Fe_{II}, %	0·1	24·0	23·4
Gangue, %	0·07	...	0·08
O_2 (remainder), %	30·02	27·58	27·68

These figures are very close to the stoichiometric compositions of the respective compounds. Oxygen was controlled by reduction with H_2, and full agreement was obtained. The specific gravity was determined by weighing in toluol of *pro analysi* grade. The mean values for the purest crystals were 5·261 ± 0·001 g./c.c. for Fe_2O_3, and 5·136 ± 0·001 g./c.c. for Fe_3O_4.

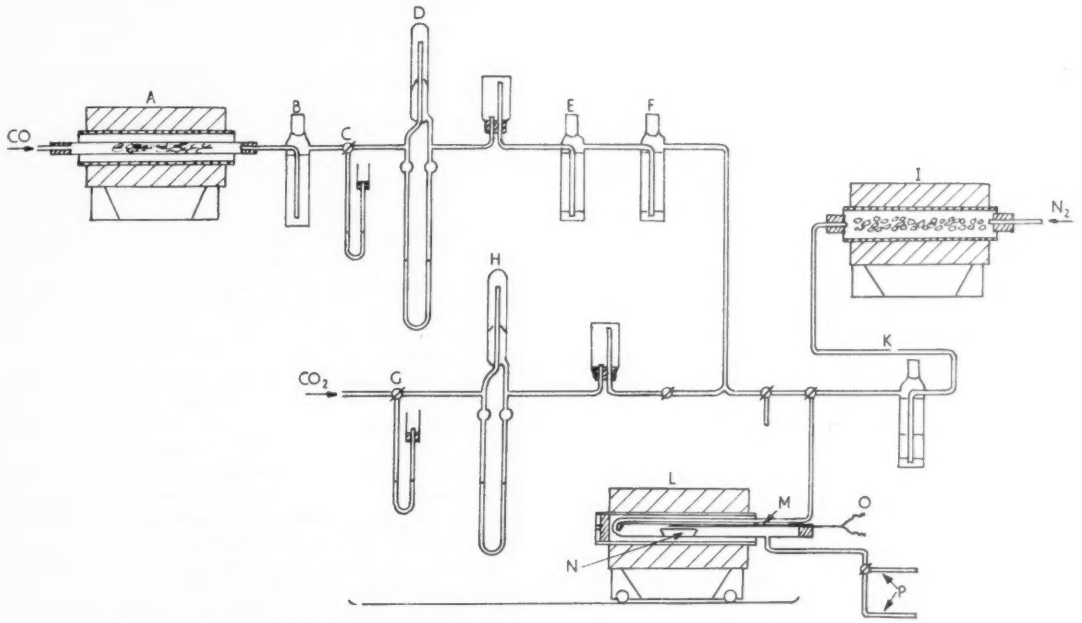
Apparatus and Method

The reduction was carried out in a bent quartz tube (Fig. 3), as used by Wiberg,[1] and was heated in an electric tube furnace. With the shape of quartz tube employed, the cooling could be easily regulated.

To obtain approximately the same size of surface on every sample, cubes with about 4-mm. edge were prepared by grinding. For each experiment two cubes were weighed. The sample was inserted in the tube on a boat fired to constant weight. At the lower temperatures the boat was of quartz, at higher temperatures, alundum. The temperature was hand-regulated by transformer and could be kept constant within ± 2° C.

The heating to reaction temperature was done very cautiously in a N_2 atmosphere and with stepwise pushing of the furnace to avoid cracks in the crystals. The cooling, also in N_2, could be done fairly quickly without causing any cracking of the reduced tubes, at least when metallic iron had been obtained. If the furnace was pulled away entirely, the time for cooling from 1000° to 500° C. was about 2 min.; after 5 min. the temperature was 250–300° C. This cooling speed avoided eutectoid decomposition of the wüstite. Pre-eutectoid magnetite was, however, precipitated.

The H_2 used was purified from O_2 by catalytic



A—Carbonyl decomposition furnace
B—Tower with glass wool
C—Mercury manometer
D—Flowmeter
E—Absorption tower with ascarite
F—Absorption tower with dehydrite
G—Mercury manometer
H—Flowmeter
I—Nitrogen-purifying furnace with copper precipitated on porous brick
K—Absorption tower with ascareti and dehydrite
L—Reduction furnace with Kanthal heating element
M—Silica reduction tube
N—Silica boat with specimen
O—Thermocouple
P—To CO_2 absorption bottles or gas analysis apparatus

Fig. 3—Diagram of CO reduction unit

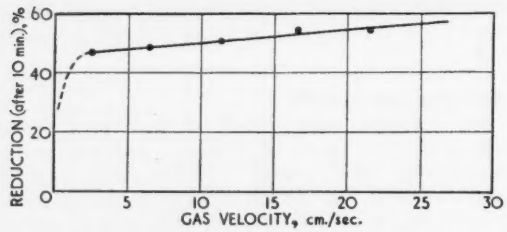


Fig. 4—Effect of gas velocity on rate of reduction of hematite with CO at 1000° C.

combustion on Pd sponge and by freezing out of the water formed. It contained, on delivery, less than 0·001% of O_2. The CO was prepared from formic acid and sulphuric acid and was subjected to after-drying, being compressed in steel tubes. This gas was also very clean, but contained small quantities of carbonyls, which had to be decomposed before use.

The CO was purified as shown in Fig. 3, which also illustrates the devices for obtaining predetermined compositions of CO–CO_2 gas mixtures. The apparatus used in the H_2 reduction experiments was similar to that used with CO. The partial pressure of H_2O was regulated by means of a saturator, working at a known constant temperature.

When pure H_2 was used as reducing agent, the reaction was followed by alternating weighing of two absorption bottles, where the H_2O formed was collected. The oxygen removed from the oxides could then be directly calculated from the weighed H_2O. With CO it was impossible to calculate the oxygen removed directly from the absorbed CO_2 weight. During reduction a carburization of the reduced iron is effected, as well as precipitation of free carbon, causing formation of CO_2:

$$2CO \rightleftharpoons CO_2 + C.$$

It was thus necessary to determine the loss of weight of the sample, and from this and the absorbed CO_2, to calculate the O_2 removed. At 1000° C. and higher, and with Fe_2O_3 as the original material, the carbon precipitation is so insignificant at first that curves could be obtained by alternating weighing of absorption bottles until about 80% of the oxygen had been removed.

From the calculated O_2 removed, the 'reduction percentage' was calculated and plotted against the time. The 'reduction percentage' is oxygen removed expressed as a percentage of the total available as iron oxides in the original material.

The influence of gas velocity was studied in the reduction of Fe_2O_3 with CO at 1000° C. The limited supply of suitable oxides prevented the extension of these experiments to other temperatures and to other raw materials and gas compositions, but the influence of these factors is fairly well known. In Fig. 4 oxygen removed after 10-min. reduction of 4-mm. Fe_2O_3 cubes has been plotted against gas velocity. The threshold value above which the gas velocity has a very small influence on the rate of reduction is very low. Subsequent tests have been performed at a constant mass velocity of 9 cu. dm./sq. cm./hr., which at 1000° C. corresponds to a linear gas velocity of 12 cm./sec. This velocity was used by Wiberg[1] in reduction tests on magnetite cubes.

Preparation of Samples for Microscopic Investigation

The grinding and polishing of a partially reduced iron oxide sample presents certain problems. The original phases, Fe_2O_3 and Fe_3O_4, are hard and brittle, whereas the layers of wüstite and iron formed are very soft and malleable and more or less porous.

Reduction pores and cracks were filled with Bakelite solution, strongly diluted with acetone to obtain sufficient fluidity, using a test tube that could be evacuated by water suction. During evacuation the sample and the Bakelite solution were kept separate and after 10–20 min. the sample was immersed in the liquid, under vacuum. The test tube was then placed in a heating chamber where the solution was evaporated at 80–90° C., so that no gas evolution was obtained inside the liquid. The evaporation took 15–20 hr. The Bakelite was hardened at 115–130° C. in 2–3 hr. The test tube was broken and the lump with the sample was then either ground as obtained or reshaped in a Bakelite press. This gives very good filling of the pores. By suitable adjustment of the hardening of the Bakelite, a good armouring of the porous and soft reduction products is obtained, which is necessary for the preparation of a satisfactory micro sample. Other filling agents, such as resin or methyl-metaakrylate, were not so suitable.

The cast sample was ground and polished as follows:

(1) Wet grinding on a rapid horizontal grinding disc. The abrasive grains were fixed on the surface of the disc with the same type of Bakelite as used for filling

(2) Polishing on cloth with diamond paste. Grain size 1–2μ

(3) Final polishing with diamond paste. Grain size 0·5μ.

With the usual methods of preparing samples the results were not as good as with the method described. However, the method gives a certain relief if polishing is continued too long, and the final polishing must be very short. Diamond paste does not completely remove scratches, but this disadvantage was preferable to the relief obtained when using tonerde. The first polishing takes 20–30 min., and the sample is then still not free from grooves. Emery paper is not suitable, as the samples are broken up by coarse paper and fine paper produces relief.

Hydrochloric acid of different concentration in alcoholic solution was used for etching, and occasionally concentrated hydrochloric acid or acid stannous chloride solution diluted with alcohol.

The best method of preparation would probably be the Graton-Vanderwilt machine, where the samples are automatically ground and polished on lead discs, thinly coated with a grinding agent. A similar, and perhaps even better, machine is manufactured by Dürener Maschinenfabrik und Eisengiesserei. This type was not available. An older, Vanderwilt-type machine was tested with suspensions of diamond powder and gave very good results and the best freedom from relief, but was extremely time-consuming.

Reduction by Pure CO

Reduction experiments with CO were carried out

at 800°, 900°, 1000°, and 1100° C. Below 800° C. the precipitation of free carbon is very disturbing. With the apparatus used, temperatures above 1100° C. were not possible.

Reduction Curves—Figure 5 shows the differences in rate of reduction at 1000° C. of Fe_2O_3, Fe_3O_4, and partly oxidized Fe_3O_4. At the same temperature, gas velocity, and sample quantity, Fe_2O_3 was completely reduced after 2 hr., while not more than 80% oxygen was removed from pure Fe_3O_4 after 8 hr. If the Fe_3O_4 crystals are first oxidized, the reducibility is immediately increased, and after complete oxidation it is as high as for natural Fe_2O_3. The different character of the curves indicates a significant difference in the reduction mechanism, which will be explained by the microscopic study of the reduction products.

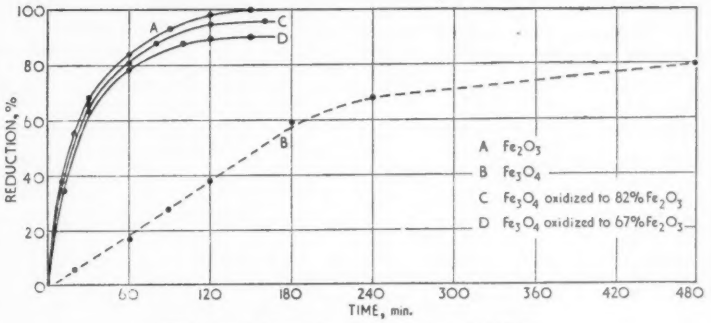


Fig. 5—Reduction with CO at 1000° C.

Temperature has little influence on the reduction velocity of Fe_2O_3 between 900° and 1100° C. (*see* Fig. 44*b*), but at 800° C. the final reduction is considerably retarded. The reduction of Fe_3O_4 is more dependent on the temperature than is the Fe_2O_3 reduction.

The reproducibility of experiments using Fe_2O_3 is good, but for Fe_3O_4 it is not very good, nor is it when the chemical composition is the same from sample to sample.

Microscopical Investigations—The structure of a sample of magnetite, partially reduced by CO at 1000° C., is shown in Fig. 6. Part of the unreduced core is seen in the lower left-hand corner. Outside this there is a relatively thick, dense wüstite layer, and in the part of this closest to the core, precipitation of pre-eutectoid Fe_3O_4 may be observed. The sample was etched with 1% HCl in alcohol, to differentiate between Fe_3O_4 and wüstite. Owing to the extent of the wüstite zone it is difficult to adjust the reduction time so that all three phases are obtained simultaneously in a 4-mm. sample. In Fig. 6, only a beginning of iron formation is seen in the upper right-hand corner. The micrograph may be said to confirm the observation of Wiberg, that the transport of matter across the wüstite layer during reduction of magnetite is a diffusion in solid solution.

Figure 7 shows a Fe_2O_3 crystal, reduced to about the same extent as in Fig. 6. In Fig. 7, as shown by the reduction curves, the reduction required 6 min., compared with 94 min. for Fig. 6. The phases present are unreduced Fe_2O_3 core, a thin, dense Fe_3O_4 layer, and dense wüstite with Fe_3O_4 precipitation. In the outer parts this phase contains elongated pores, which are close to the crystal surface lined with metallic iron. The same sample, unetched, is shown in Fig. 8, where the iron in the pores is clearly seen.

These crystals were also studied at higher magnification. Figure 9 shows the dense Fe_3O_4 layer and (to the right) wüstite with precipitated Fe_3O_4. Figure 10 shows the continuation of the wüstite layer with magnetite and (to the right) the elongated pores. On the right of Fig. 11, which shows a section closer to the surface, the partly iron-lined pores are seen. Figure 12 shows how the grain boundaries in the wüstite layer of the Fe_3O_4 crystal are indicated by the Fe_3O_4 precipitation. In the top left, portions of the magnetite are enveloped by the reaction front, especially if they contain an impurity. Such portions are also found in the interior of the sample, and similar phenomena have been observed in reduced Fe_2O_3 samples.

The grains of the thin magnetite layer in a partially reduced Fe_2O_3 crystal may, in spite of the fact that the magnetite has a cubic lattice, be seen in an etched sample under polarized light. The etched section reflects the polarized light in a way that is characteristic of the orientation of the crystal grain, as it contains etching spots with a regular system of surfaces which have a certain position in relation to the direction of radiation and the polarization plane.[56] Figure 13 shows clearly the arrangement of the grains as a single row in the magnetite layer. The magnetite is fine-grained, which increases the reactivity. The pores in the wüstite sometimes start in the magnetite grain boundaries.

Figures 14, 15, and 16 show specimens further reduced than those previously discussed. About 50–60% of oxygen has been removed. The structures of iron from Fe_3O_4 and from Fe_2O_3 are seen in Figs. 17 and 18 to be quite different in the two samples. From Fe_3O_4 the iron is finely porous, while from Fe_2O_3 it has large, elongated pores, a structure seen already in the wüstite stage. Figure 16 shows a Fe_3O_4 crystal, oxidized almost completely to Fe_2O_3 and then reduced in CO to the same degree as the crystals in Figs. 14 and 15. The reduced sample is very similar to reduced natural Fe_2O_3. If oxidation is not complete, the structure at this degree of reduction is as shown in Fig. 19. Such a sample appears to have a higher strength than those obtained from completely oxidized crystals.

The elongated porosity in a reduced Fe_2O_3 sample is finer at lower reduction temperatures. Cracking in the hematite core is also more extensive (Fig. 20). In Fe_3O_4 crystals, which appear to be less ideally built up, the boundary between the wüstite and iron layers is not so sharp as in perfect crystals. Figure 21 shows structures of such a sample reduced at

Table I

PHASES FORMED BY REDUCTION WITH CO ACCORDING TO X-RAY INVESTIGATIONS

Exposure No.	Original Oxide	Reduction Temp., °C.	Reduction Time, min.	Reduction, %	Phases Observed	Remarks
1	Fe_2O_3	...	...	...	Fe_2O_3	Original oxide
2	Fe_3O_4	...	...	...	Fe_3O_4	,, ,,
3	Fe_2O_3	1000	30	...	Fe_3O_4	Reducing agent 11% CO + 89% CO_2
4	,,	,,	8	14·5	Fe_2O_3, Fe_3O_4, FeO	Reducing agent 58% CO + 42% CO_2
5	,,	,,	2	9·6	Fe_2O_3, Fe_3O_4, FeO	Reducing agent CO
6	,,	,,	5	27·3	Fe_2O_3, FeO, Fe	Reducing agent CO
7	Fe_3O_4	,,	60	34·2	Fe_3O_4, FeO, Fe	Reducing agent CO
8	Fe_2O_3	800	4	16·5	Fe_2O_3, (Fe_3O_4), FeO, Fe	Reducing agent CO. Very weak Fe_3O_4 lines

900° C. The tendency of the structure to break up is more marked at low temperatures.

The extensive formation of pores and the breaking up of the structure at an early stage during reduction of Fe_2O_3 facilitates the continued reduction of the sample as the surface available for gas attack is enlarged and the diffusion distances in the solid phase are shortened. This is thus the most important contributing cause of the easy reducibility of hematite at this temperature, compared with the other oxides of iron, but it is also the cause of the low strength of hematite during reduction. The wüstite stage is the weakest during reduction of a hematite,[57] as would be expected from the structure of the reduction products.

The early formation of pores during the reduction of hematite overshadows the influence of cracks and other minor differences in the free surface of the specimen. This explains why the reproducibility with Fe_2O_3 is so much better than with Fe_3O_4, where differences in the original material are much more important.

X-Ray Studies—The phases present in a sample partially reduced by CO at 1000° C. were controlled by an X-ray spectrometer with a Geiger-Müller counter. The reduction was performed on lumps, pulverized before X-ray examination. Control tests show the limit for observation of Fe_3O_4 beside Fe_2O_3 to lie at about 4% Fe_3O_4, which indicates the smallest quantities of a phase identifiable with this method.

The phases are listed in Table I, which includes samples reduced with CO–CO_2 mixtures, and are those expected from the equilibrium diagram. When lump oxide is reduced by pure CO, Fe_3O_4 lines are observed only at first. When the reduction has proceeded further and the thin Fe_3O_4 layer, which throughout the process has about the same thickness, lies near the centre of the lumps and thus constitutes a very small percentage of the sample, the Fe_3O_4 phase cannot be determined by this method. It can be observed under the microscope (Fig. 7).

Reduction by CO–CO_2 Mixtures

To make a closer study of the intermediate products formed during reduction of Fe_3O_4 and Fe_2O_3, experiments were performed with CO–CO_2 gas mixtures. The equilibrium diagram Fe–C–O (Fig. 2*a*) shows how the CO/CO_2 ratio should be chosen to obtain a desired final product.

Figures 22, 23, and 24 show wüstites from Fe_3O_4, Fe_2O_3, and Fe_3O_4 oxidized to Fe_2O_3. The wüstite from Fe_3O_4 is dense, while that from Fe_2O_3 and oxidized Fe_3O_4 is coarsely porous, as in the previous experiments, but the porosity is not quite so elongated. This may be due to the fact that the sample was not completely oxidized to Fe_2O_3. Complete oxidation takes a very long time. Further hematite obtained through oxidation of a magnetite crystal is polycrystalline, but this has no very great influence on the structure (as shown later).

The difference in reducibility of the different forms of wüstite is shown in Fig. 41. The wüstites were prepared at 1000° C. with a gas mixture of 40% CO_2 and 60% CO. When Fe_2O_3 is reduced with pure CO, the thin Fe_3O_4 layer is dense (Fig. 9); but if the gas mixture is such that the whole sample is transformed to Fe_3O_4, the elongated type of porosity is obtained as in the wüstite (Fig. 25). The sample then has a reducibility comparable with that of hematite (*cf.* Fig. 45).

Comparison of Reduction by H_2 and by CO at 1000° C.

Reduction of both Fe_2O_3 and Fe_3O_4 crystals at 1000° C. appears to be faster by H_2 than by CO (Fig. 42), particularly with Fe_3O_4. Reduction of hematite by H_2 gives approximately the same structures as by CO. The porosity in wüstite is, however, as in the reduced iron, considerably finer, and the dense wüstite layer is thinner, in the former case (Fig. 26). In a rapidly cooled sample the phase boundary Fe_2O_3–Fe_3O_4 is well marked, and is often a straight line, but is sometimes broken (Fig. 27). The phase boundary Fe_3O_4–wüstite is very irregular. In the course of diffusion, islands of magnetite are enveloped behind the reduction front. The pores formed in the wüstite are orientated in the direction of the main diffusion current. In the photomicrographs illustrating reduction by CO, very few cases of enveloping are seen. This may be due to the fact that these samples were cooled very slowly and that the magnetite islands had then been dissolved, magnetite afterwards precipitating pre-eutectoidally.

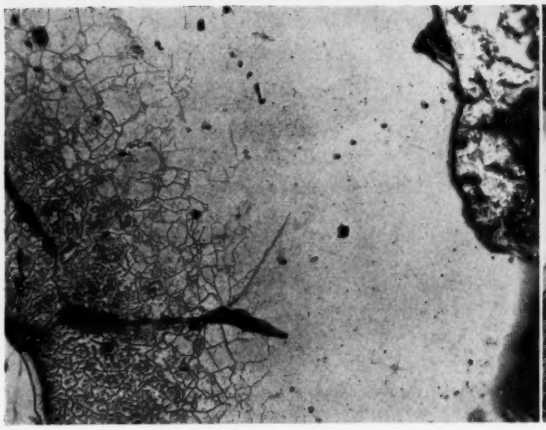


Fig. 6—Magnetite crystal showing magnetite, wüstite, and metallic iron: 1000° C., CO, 94 min., etched × 45

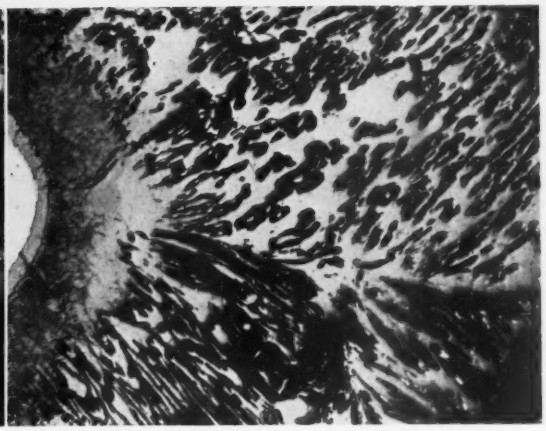


Fig. 7—Hematite crystal showing hematite, magnetite, dense wüstite, and porous wüstite with iron-lined pores (in outer parts of specimen) : 1000° C., CO, 6 min., etched × 60

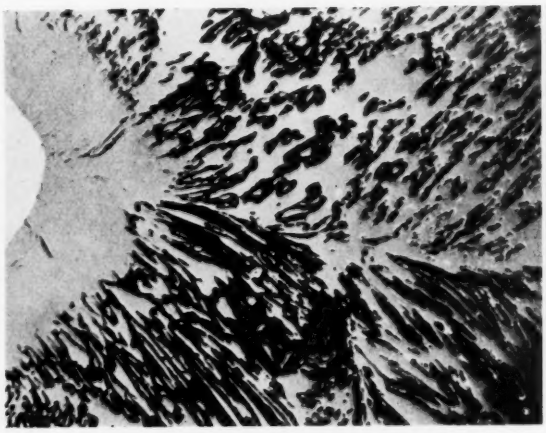


Fig. 8—As Fig. 7, unetched specimen

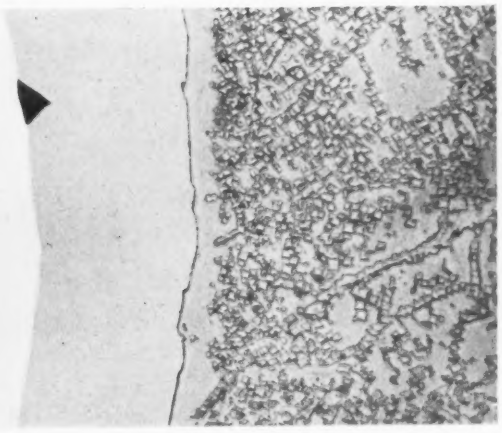


Fig. 9—Hematite crystal showing hematite, magnetite, and wüstite with precipitated magnetite: 1000° C., CO, 6 min., etched × 600

Fig. 10—Hematite crystal showing wüstite with magnetite and pores: 1000° C., CO, 6 min., etched × 600

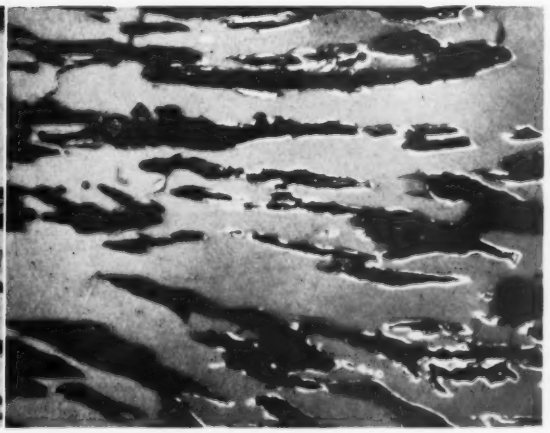


Fig. 11—Hematite crystal showing wüstite with partly iron-lined pores : 1000° C., CO, 6 min., unetched × 600

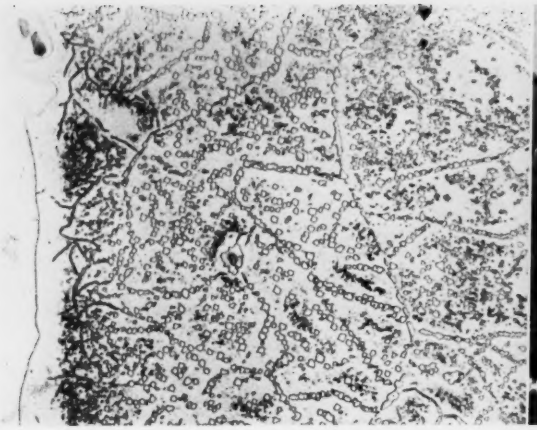


Fig. 12—Magnetite crystal showing magnetite and wüstite : 1000° C., CO, 94 min., etched × 200

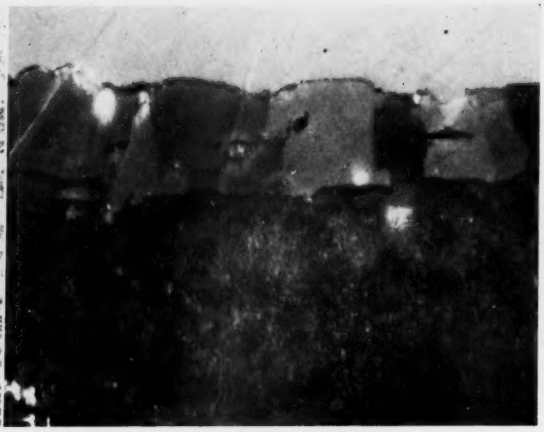


Fig. 13—Hematite crystal showing hematite, magnetite, and wüstite : 1000° C., CO, 6 min., etched × 300

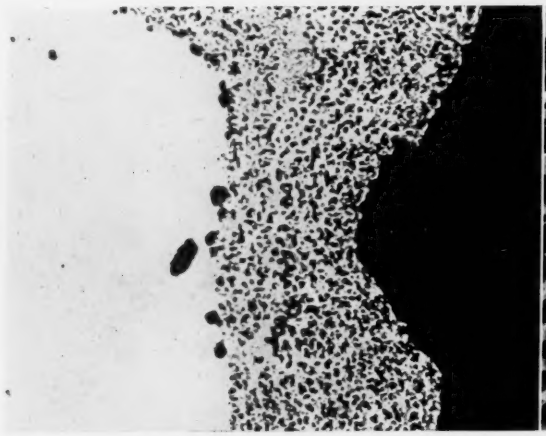


Fig. 14—Magnetite crystal showing wüstite and metallic iron : 1000° C., CO, 140 min., unetched × 200

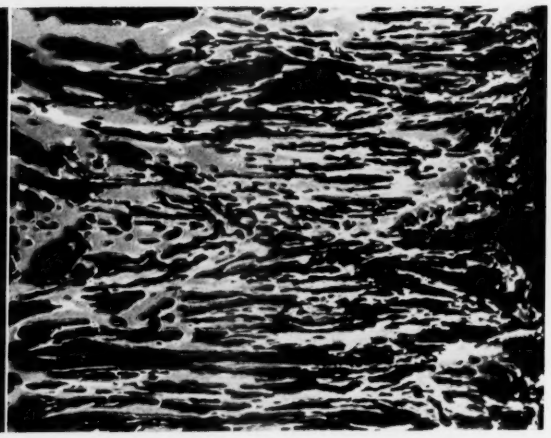


Fig. 15—Hematite crystal showing wüstite and metallic iron : 1000° C., CO, 10 min., unetched × 200

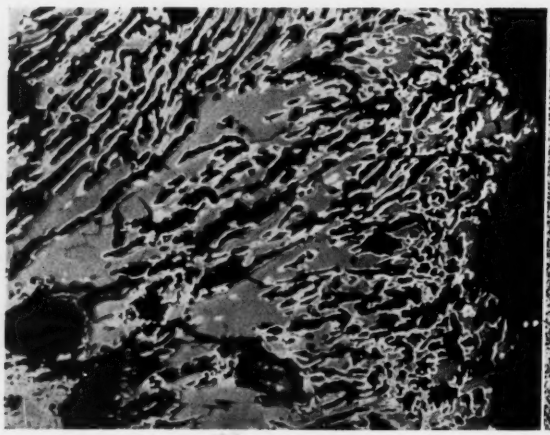


Fig. 16—Magnetite crystal oxidized to hematite and reduced, showing wüstite and metallic iron : 1000° C., 8 min., unetched × 200

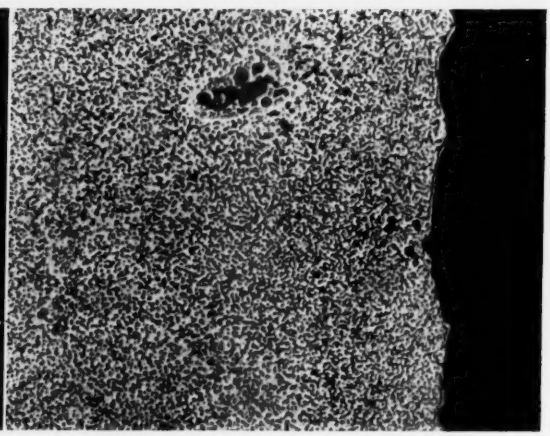


Fig. 17—Magnetite crystal showing metallic iron (outer part of specimen) : 1000° C., CO, 480 min., unetched × 200

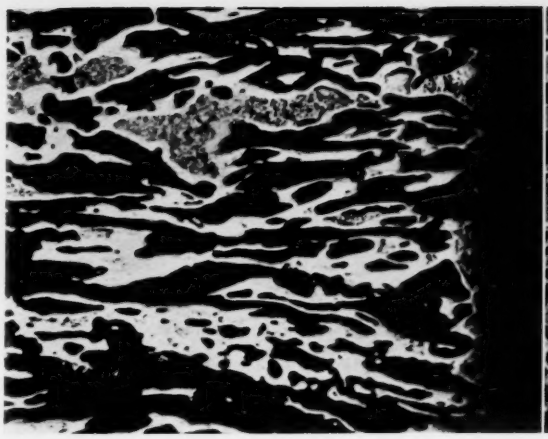


Fig. 18—Hematite crystal showing metallic iron : 1000° C., CO, 155 min., unetched × 200

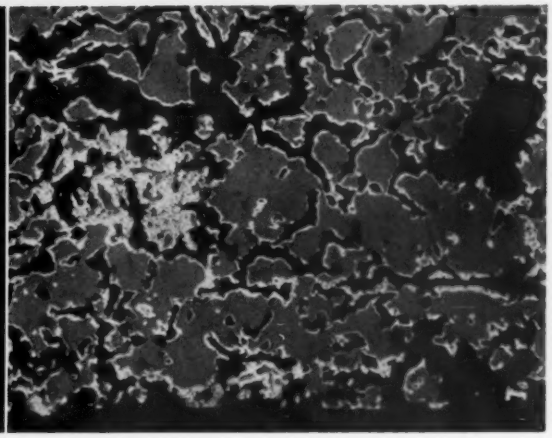


Fig. 19—Magnetite crystal oxidized to 82% hematite and reduced, showing wüstite and metallic iron : 1000° C., CO, 12 min., unetched × 200

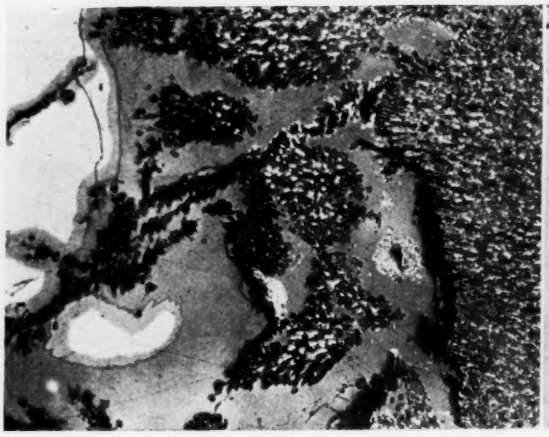


Fig. 20—Hematite crystal showing hematite, magnetite, wüstite, and metallic iron : 800° C., CO, 30 min., etched × 120

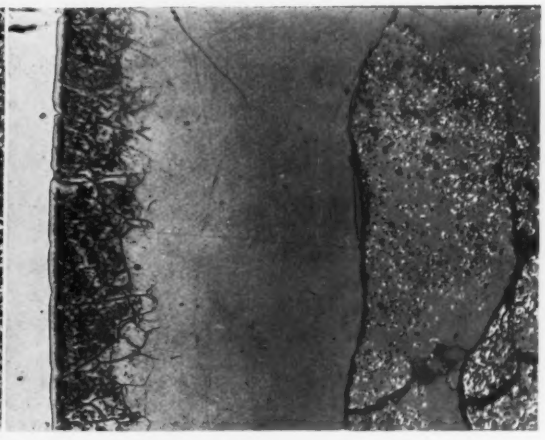


Fig. 21—Magnetite crystal showing magnetite, wüstite, and metallic iron : 900° C., CO, 210 min., etched × 120

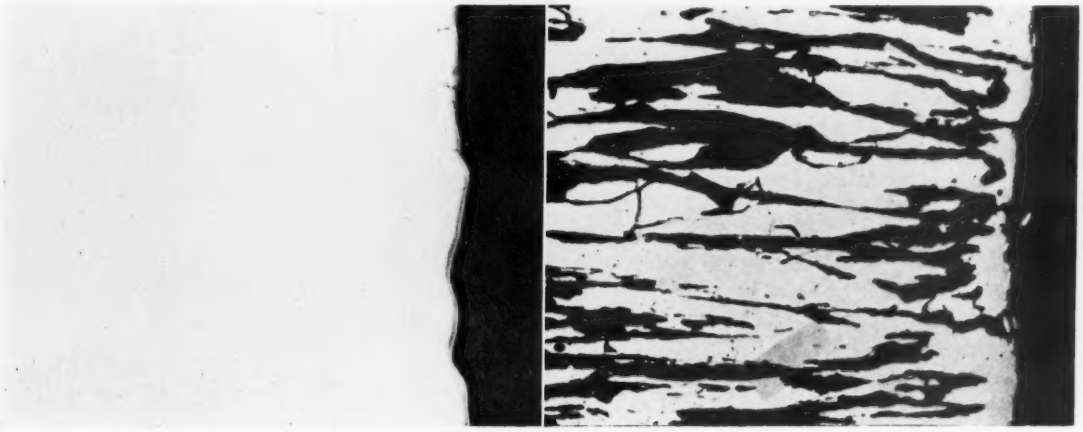


Fig. 22—Magnetite crystal showing dense wüstite : 1000° C., 60% CO + 40% CO_2, 330 min., unetched × 200

Fig. 23—Hematite crystal showing porous wüstite : 1000° C., 60% CO + 40% CO_2, 210 min., unetched × 200

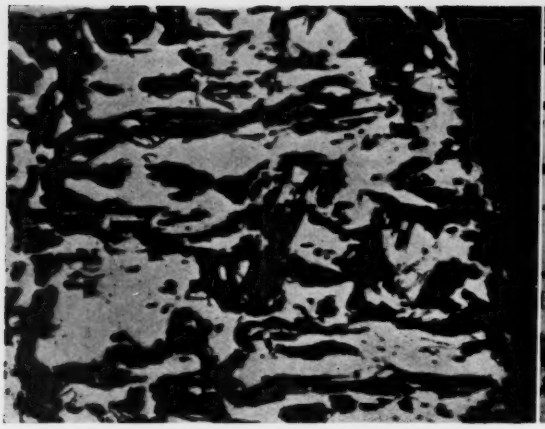


Fig. 24—Magnetite crystal oxidized to hematite and reduced, showing porous wüstite : 1000° C., 60% CO + 40% CO_2, 120 min., unetched ×200

Fig. 25—Hematite crystal showing porous magnetite : 1000° C., 89% CO_2 + 11% CO, 360 min., unetched × 200

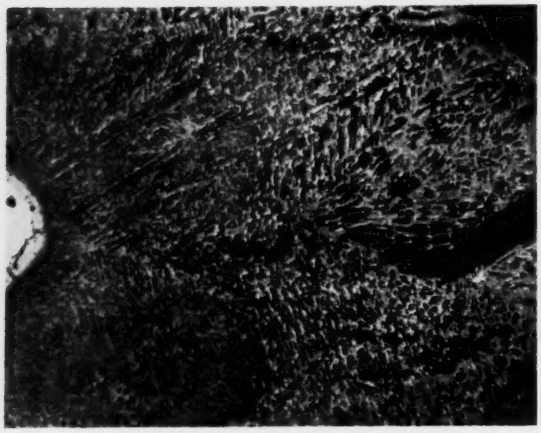


Fig. 26—Hematite crystal showing hematite, magnetite, dense wüstite, and porous wüstite with iron-lined pores (outer parts of specimen) : 1000° C., H_2, 2 min., etched × 60

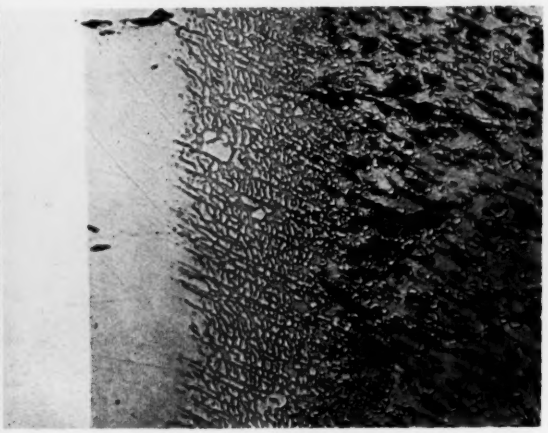


Fig. 27—Hematite crystal showing hematite, magnetite, and wüstite with magnetite : 1000° C., H_2, 2 min., etched × 600

Fig. 28—Magnetite crystal showing magnetite, wüstite, and metallic iron : 1000° C., H_2, 6 min., etched × 60

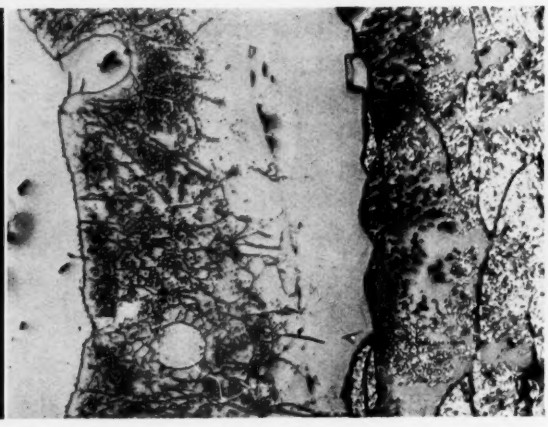


Fig. 29—Magnetite crystal showing magnetite, wüstite, and metallic iron : 1000° C., H_2, 6 min., etched × 200

Fig. 30—Hematite crystal showing hematite, magnetite, wüstite, and metallic iron : 700° C., H_2, 8 min., etched × 60

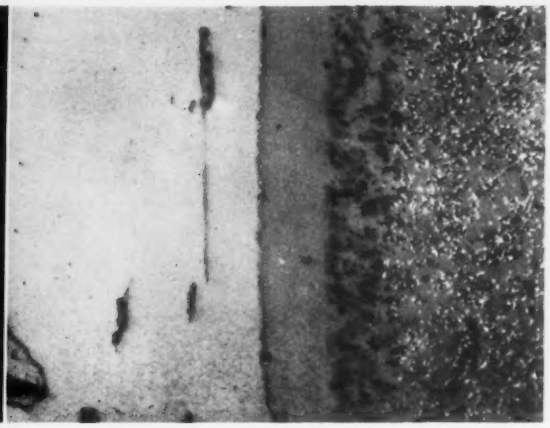


Fig. 31—Magnetite crystal showing magnetite, wüstite, and metallic iron : 700° C., H_2, 20 min., etched × 600

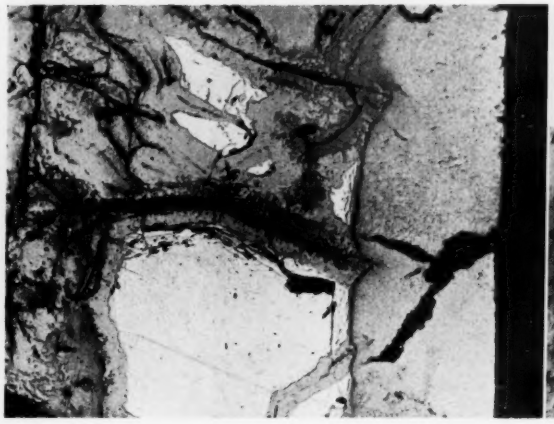


Fig. 32—Hematite crystal showing hematite, magnetite, wüstite, and metallic iron : 600° C., H_2, 16 min., etched × 60

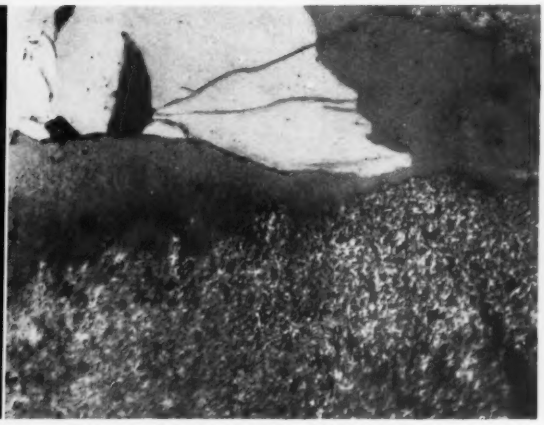


Fig. 33—Hematite crystal showing hematite, magnetite, wüstite, and metallic iron : 600° C., H_2, 16 min., etched × 600

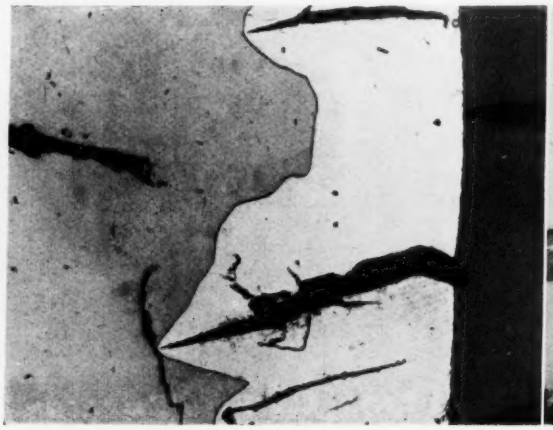


Fig. 34—Magnetite crystal showing magnetite and metallic iron : 600° C., H_2, 35 min., etched × 60

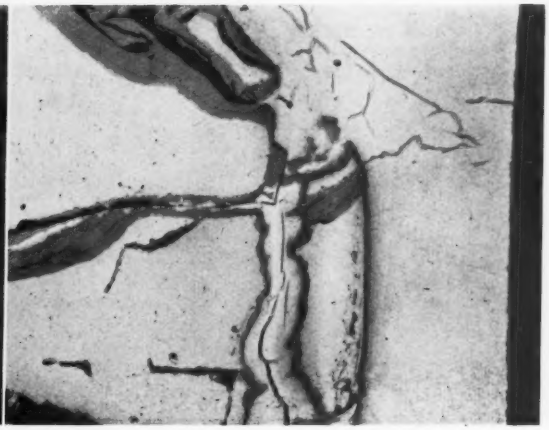


Fig. 35—Hematite crystal showing hematite, magnetite, and metallic iron : 450° C., H_2, 170 min., etched × 60

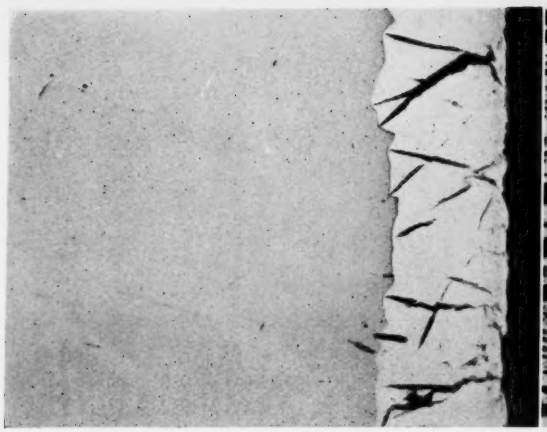


Fig. 36—Magnetite crystal showing magnetite and metallic iron : 450° C., H_2, 120 min., unetched × 60

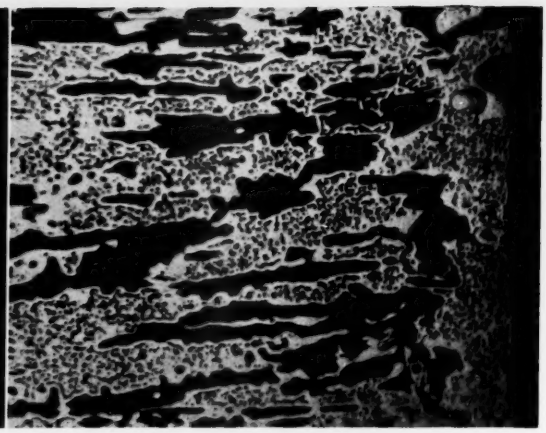


Fig. 37—Hematite crystal reduced after preliminary reduction to magnetite showing metallic iron : 1000° C., CO, 180 min., unetched × 200

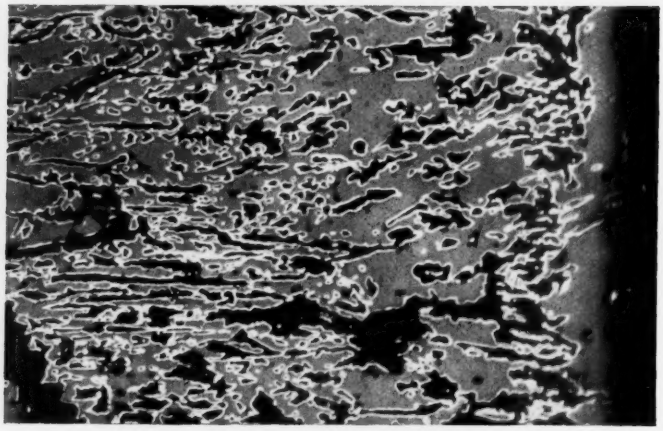


Fig. 38—Hematite ore showing wüstite and metallic iron : 1000° C., CO, 8 min., unetched × 200

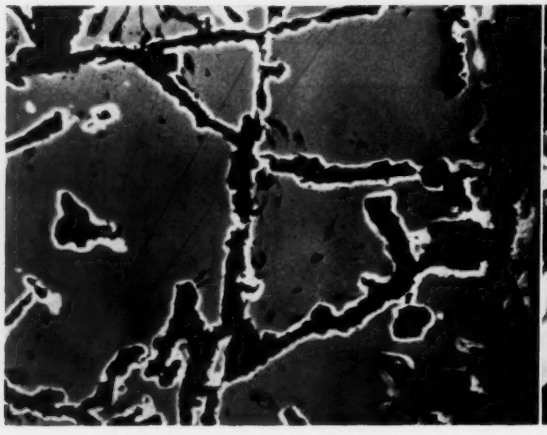


Fig. 39—Magnetite ore showing wüstite and metallic iron : 1000° C., CO, 65 min., unetched × 200

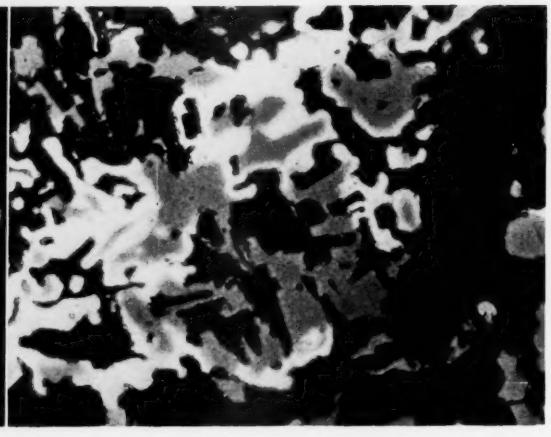


Fig. 40—Hematite pellets showing wüstite and metallic iron : 850° C., CO, unetched × 600

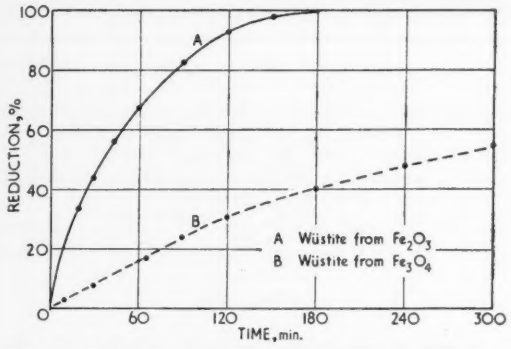


Fig. 41—Reduction of wüstite with CO at 1000° C.

The microstructures of the magnetite reduction products do not correspond to the same extent with the two reducing agents. With H_2 there is a tendency to form pores in the wüstite (Figs. 28 and 29), and the wüstite–iron boundary is very irregular. The dense wüstite zone is rather thin and no pronounced iron zone appears, but, as with hematite, wüstite alternates with iron in the external parts of the sample. The structure of the dense wüstite layer is similar to that with CO. Figure 29 shows clearly how the larger magnetite islands arise by envelopment.

When reducing Fe_2O_3 and Fe_3O_4 by H_2, and especially the latter, a small zone of dense wüstite and larger reaction surfaces between wüstite, iron, and gas are obtained, which, together with the higher diffusion velocity of H_2, probably causes a faster reaction than with CO.

Temperature Dependence of Reduction by H_2

Reduction Curves—The difference in reduction rate of Fe_2O_3 and Fe_3O_4 by H_2 at 1000° C. is not so great as by CO (Fig. 42). The difference is, however, considerable, and it persists at lower temperatures until slightly below the wüstite point (Fig. 43). The reduction rate at 450° C., *i.e.*, well below the stability range of wüstite, seems to be determined by the phase-boundary reaction of magnetite to iron. The curves are linear and the difference in reactivity has disappeared.

From similar curves for temperatures between 1000° and 450° C. the times necessary to reach 40, 70, and 95% reduction were determined. Diagrams were drawn to show the temperature dependence of these times (Fig. 44). Available data for CO were also included.

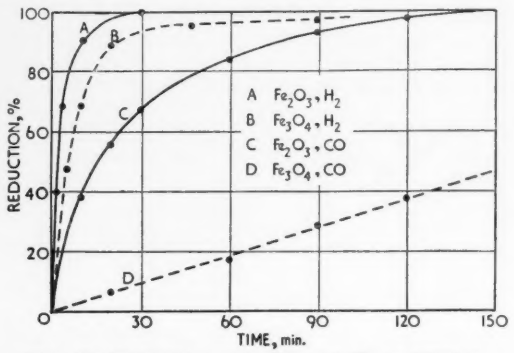


Fig. 42—Reduction with CO and H_2 at 1000° C.

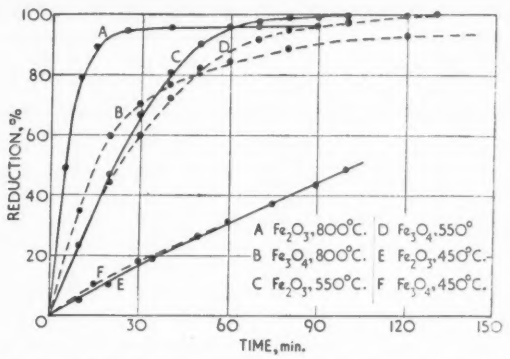


Fig. 43—Effect of temperature on reduction of hematite and magnetite with H_2

The curves show clearly, for Fe_2O_3 and Fe_3O_4, that reduction by H_2 has a pronounced minimum velocity slightly above the temperature where wüstite (according to the equilibrium diagram) may be expected to appear. The effect is more marked in Fe_3O_4, where the temperature must be above 850° C. to reach 95% reduction in the same time as at 550° C. The final reduction, particularly, is retarded, and the diagrams indicate different reaction mechanisms above and below the minimum.

The Fe_3O_4 curves show a discontinuity at about 900° C., as in previous investigations. The reduction to γ-Fe seems to be somewhat retarded in comparison with that to α-Fe. No clear discontinuity was observed in Fe_2O_3 reduction, but the rise in reactivity above 900° C., as for CO, is very slight.

Microstructures—The structures at 1000° C. have already been described. At lower temperatures the porosity in the wüstite from Fe_2O_3 is finer and the magnetite has a more irregular appearance, as in Fig. 30, which shows the structure at 700° C. The dense wüstite zone in a partially reduced Fe_3O_4 sample is very thin at this temperature (Fig. 31), and iron and wüstite are formed as an intimate mixture. The difference in Fe_2O_3 and Fe_3O_4 samples in the wüstite–iron zone is therefore very slight in this instance. Wüstite is easily detected after etching, in both Fe_2O_3 and Fe_3O_4 samples reduced at temperatures down to 650° C.

At 600° C. magnetite from Fe_2O_3 has a still more irregular form (Fig. 32). The crystal cracks during reduction and magnetite and wüstite are formed in these cracks, thus dividing the hematite into numerous smaller grains. This tendency, as with CO, is already evident at 800° C. The wüstite in the cracks forms relatively thick and dense layers. Iron is seldom found in the cracks, but is formed in an even border layer around the sample and contains, in turn, big cracks in some places. Below 600° C. the iron reduced is pyrophoric. Oxidation may be prevented by pouring acetone over the sample immediately after removal from the furnace or by heating the reduced sample for a short time at 650–700° C.

The boundary towards the iron phase often consists, according to the microscopic investigation, of a direct

Table II—PHASES FORMED BY REDUCTION WITH H_2 (X-RAY INVESTIGATIONS)

Exposure No.	Original Oxide	Reduction Temp., °C.	Reduction Time, min.	Reduction, %	Phases Observed
1	Fe_2O_3	700	15	63·1	Fe_2O_3, Fe_3O_4, FeO, Fe
2	Fe_3O_4	,,	,,	14·8	Fe_3O_4, FeO, Fe
3	Fe_2O_3	650	35	79·2	(Fe_2O_3), Fe_3O_4, FeO, Fe
4	Fe_3O_4	,,	,,	64·8	Fe_3O_4, FeO, Fe
5	Fe_2O_3	600	20	45·7	Fe_2O_3, Fe_3O_4, FeO, Fe
6	Fe_3O_4	,,	,,	50·0	Fe_3O_4, Fe
7	Fe_2O_3	,,	30	60·1	Fe_2O_3, Fe_3O_4, FeO, Fe
8	Fe_3O_4	,,	,,	62·7	Fe_3O_4, Fe
9	Fe_2O_3	550	20	28·4	Fe_2O_3, Fe_3O_4, Fe
10	Fe_3O_4	,,	,,	37·2	Fe_3O_4, Fe

transition from magnetite to iron without an intermediate wüstite layer, and the cracks in the iron generally start from this boundary (Fig. 33). Reaction surfaces between Fe_3O_4 and iron are orientated parallel to the external surface of the specimen and are thus perpendicular to the strongest mass flow.

In a Fe_3O_4 crystal, partially reduced at 600° C., no wüstite could be seen even after etching in concentrated acids. The magnetite appears to transform directly to iron (Fig. 34). The structure is very similar to the one appearing below the eutectoid point of wüstite (*cf.* Fig. 36). The iron phase is finely porous, but with big cracks. Figure 35 shows the structure of an Fe_2O_3 crystal reduced so that no wüstite is formed.

These microscopic findings were confirmed by X-ray crystallographic methods, as shown in Table II. The wüstite obtained in the reduction of Fe_2O_3 by H_2 at 600° C. has a lattice parameter, which, according to the curve of Jette and Foote,[19] indicates a considerably higher iron content than is found in the equilibrium range. The lattice parameter decreases with increasing temperature.

From the foregoing phase studies, there is good reason to suppose that the reaction minimum at 625–800° C. is largely caused by the formation, in this temperature range, of a continuous wüstite layer, across which diffusion proceeds very slowly. At lower temperatures, where wüstite does not appear, a cracked and finely porous iron is obtained, and thus the magnetite surface is always free for new action of the gas. The recrystallization and sintering of reduced iron, which starts at about 600° C., may contribute to the retardation of the reaction.[40, 42, 43]

There may be several co-operating causes of the irregular appearance of wüstite and also of magnetite in samples reduced at 600° C. As seen from Fig. 32, magnetite layers perpendicular to the reduction front are very thin where wüstite does not appear in a partially reduced Fe_2O_3 sample. Inside the cracks in the core, where wide wüstite layers are present, there is no iron, and the reduction potential is low, magnetite always forms relatively wide layers. The reduction of Fe_2O_3 to Fe_3O_4 is slightly exothermic, whereas the reduction of Fe_3O_4 or wüstite to iron is strongly endothermic. The reaction in the core cracks

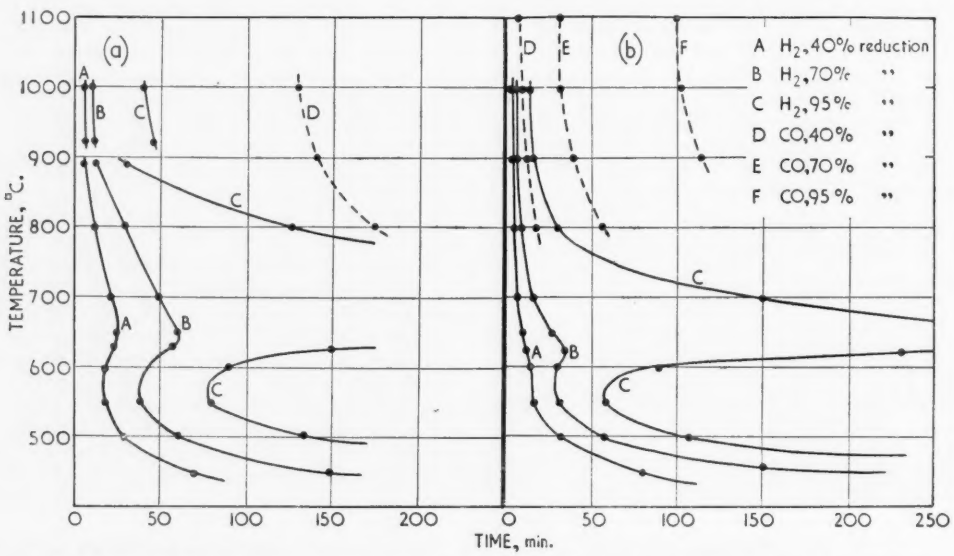


Fig. 44—Effect of temperature on reduction of (*a*) magnetite, and (*b*) hematite, with H_2 and CO

should thus tend to raise the temperature, while the reaction at the boundary towards the iron phase has a cooling influence. However, it is hardly probable that such a large temperature difference between surface and core could alone explain the formation of wüstite in the core but not at the surface.

If the wüstite layer is very thin at the reaction temperature, this phase might be divided between the surrounding layers of magnetite and iron during cooling. Studies of the reaction between iron and magnetite have shown, however, that reaction in the solid phase at these temperatures is very slow. At 600° C. about ½-hr. annealing is necessary to obtain sufficient wüstite at a boundary surface between magnetite and iron to be observed in the microscope.[58] The crystals in Fig. 32 were cooled in less than 2 min.

The deciding factor appears to be that, at temperatures around 600° C., the mobility in the solid phase is so limited that, when magnetite is reduced, the phase-boundary reaction goes directly to iron without any solid-phase diffusion. When hematite is reduced, the reaction goes directly, provided that high partial pressures of H_2O in the gas do not retard the reaction, in which case wüstite may be formed.

Changes in Dimensions due to Reduction

In reduction of iron oxides by CO or H_2 the external volume of the sample often changes considerably. Wiberg[1] found that when cubes of Kiruna A-ore (a magnetite) were reduced, a shrinkage took place, while reduction with CO first caused a relatively insignificant swelling which later decreased. The temperature was 1000° C. Similar observations were made by Tenenbaum and Joseph.[45, 48] In the present investigation the changes in dimensions were investigated, as they are closely associated with structural changes in the sample. The strength of the ore during reduction, which is of course greatly dependent on the structure, seems also to be related to the volume increase. The compressive strength is inversely proportional to the swelling of the sample.[59]

Reduction of a Fe_2O_3 crystal is at first accompanied by a considerable swelling, when the characteristic porous structure is formed. Towards the end the sample shrinks again, but the final result is a considerable swelling. This applies to reduction by CO and H_2, but CO gives the larger volume increase. With magnetite, the final result is always a shrinkage.

The volume changes observed during stepwise reduction of 4-mm. cubes of Fe_2O_3 and Fe_3O_4 in CO or CO–CO_2 mixtures were as follows:

Volume Changes in Reduction by CO at 1000° C.

Original Oxide Fe_2O_3						
	Phase:	Fe_2O_3		Fe_3O_4	Wüstite	Fe
Apparent volume*		100	→ 125	→ 132	→	127
,,	,,	100	→ 125	→		125
,,	,,	100	→	138	→	126
,,	,,	100	→	→	→	148
Original Oxide Fe_3O_4						
Apparent volume*			100	→		96†

* Percentage of original volume
† Only reduced to 73%

The reduction potential of the gas was adjusted so that the crystals were transformed to the desired end-product, as described above. The arrows indicate the various reduction steps.

Wüstite cubes reduced from Fe_3O_4 were also measured, although, owing to cracks, the results are uncertain. However, the external volume changed very little in this reduction stage.

The foregoing figures show that the swelling is greater when the sample is reduced by a gas with a higher reduction potential and therefore with a high reaction rate, than with a more cautious, stepwise reduction. The swelling is also associated with reduction steps that involve diffusion in the solid phase. In reduction of wüstite to iron, where no appreciable diffusion may be supposed to occur, there is a decrease in the apparent volume. A curve showing the swelling as a function of the time during reduction of Fe_2O_3, as well as of Fe_3O_4, by pure CO at 1000° C. will therefore have a maximum. At first, when there is diffusion in solid phase, the swelling tendency dominates, but towards the end of the reduction, when only wüstite is transformed to iron, there is a shrinkage. The final result may be a swelling or a shrinkage—the former with Fe_2O_3 and the latter with Fe_3O_4. This fundamental appearance of the curves has been confirmed by Beckius and Hedman,[60] who studied the volume changes as a function of the reduction time, with a continuously working dilatometer. The curves obtained often had a parabolic shape during the swelling stage.

Reduction by H_2 agrees, on the whole, with reduction by CO. In reducing Fe_2O_3 to iron at 1000° C. an increase in apparent volume of 40% is obtained. At lower temperatures the apparent volume decreases, and at 600° C. the sample shrinks about 2%. When magnetite is reduced the final result is a shrinkage of 4–5% at 1000° C., and at 600° C., where wüstite does not appear, about 14%.

All these changes in dimensions have been measured by micrometer before and after reduction of carefully ground cubes. Samples were chosen in which the external form was well preserved during reduction. The spread of the results is, however, fairly large from test to test.

The considerable swelling of the reaction products when Fe_2O_3 is reduced causes more porosity to appear in these products than the differences in specific volume would suggest. The reduction of Fe_2O_3 to wüstite would, if the external volume were kept unchanged, give a porosity of about 16% in the wüstite. Because of the swelling the real porosity is about 38%, a figure which was also controlled by the paraffin method of Tikkanen[40] for measuring the apparent volume. The real porosity in magnetite from Fe_2O_3 is about 20%.

Mechanism of Reduction of Fe_2O_3 and Fe_3O_4

On the basis of the studies of microstructures and of the reaction $Fe + Fe_3O_4 = 4FeO$,[59] and with due consideration to previous knowledge, the author believes that, when all phases are formed, the mechanism of reduction of an ideal magnetite lattice at 1000° C. is probably as follows:

(1) Breakdown of oxygen lattice and formation of iron phase at the wüstite–iron phase boundary, or

at the boundary between wüstite, iron, and gas if the iron is sufficiently porous or cracked

(2) Diffusion of iron across the wüstite layers towards the magnetite core

(3) Migration of iron into the magnetite at the magnetite–wüstite phase boundary during formation of wüstite phase and without any considerable change of oxygen lattice.

In (3) the transformed part of the magnetite should swell somewhat if the mechanism proposed here is correct. In (1), H_2 should pass easily through the iron layer formed, if this is not too thick, and a bursting action on these layers should occur, due to collection of water vapour formed as a reaction product and enclosed by the iron, which acts as a semi-permeable membrane.[1, 40] In reduction by CO, the iron layer is carburized and is burst by a reaction between dissolved carbon and wüstite, which may give rise to very high pressures.[1] At temperatures too low for the formation of wüstite, only (1) should appear and the magnetite should transform directly to iron. As the iron phase is finely porous and very cracked, the oxide phase is laid free for new gas action (Fig. 36).

When hematite is reduced, (1), (2), and (3) should be the same as in the Fe_3O_4 reduction, except that a considerable porosity is obtained in the wüstite layer (Fig. 6) and (1) and (2) therefore proceed faster on account of a larger reaction surface and shorter diffusion distances. The following additional steps occur:

(4) Diffusion across a dense magnetite layer

(5) Phase-boundary reaction Fe_2O_3 to Fe_3O_4 accompanied by transformation of oxygen lattice from hexagonal to cubic close-packed.

There is thus a diffusion, on the one hand, across a partly porous wüstite layer and, on the other across a magnetite layer with a similar cubic close-packed oxygen lattice, formed through rearrangement of the hexagonal close-packed hematite lattice. The magnetite layer is so thin that the diffusion velocity across this may cover the change of composition from Fe_2O_3 to Fe_3O_4.

That the dense part of the wüstite layer has sufficient transportation capacity for the iron that has to migrate into the magnetite to transform this to wüstite has been verified experimentally in the above-mentioned studies of the reaction

$$Fe + Fe_3O_4 = 4FeO.$$

By measuring under the microscope the thickness of the wüstite layer, which is formed at a phase-boundary surface between iron and magnetite after annealing for a certain time in a neutral atmosphere at constant temperature, the following relation was found between layer thickness (p mm.) and annealing time (t hr.) at 1000° C.:

$$p^2 \cong 0{\cdot}5t.$$

The reaction is thus governed by the diffusion, and the increase in thickness of the layer per unit of time (dp/dt) is approximately $0{\cdot}25/p$.

When Fe_2O_3 is reduced by CO at 1000° C. the dense wüstite layer is, on an average, about 0·07 mm., and thus $dp/dt \cong 4$ mm./hr., which corresponds approximately to the velocity with which the magnetite–wüstite phase boundary surface moves towards the interior of the sample. The reduction of Fe_2O_3 by H_2 gives such thin wüstite layers that the corresponding penetration velocity is 10–15 mm./hr., which also covers the reduction velocity. The reduction of Fe_3O_4 to wüstite should therefore mainly occur in the solid phase. But a certain amount of oxygen may also be removed by gas in the pores of wüstite, where —at least when Fe_2O_3 is reduced by H_2—many enveloped Fe_3O_4 particles are in contact with the reducing gas.

The opinion has been expressed that there are many indications that the reduction should not, like the oxidation, proceed in the solid phase. It has been pointed out that in oxidation only oxygen need be added, whilst in reduction, besides the gaseous reducing agent, there is a removal of gaseous reduction products from the reduction surfaces. Further, the rate of reduction should be too rapid for diffusion across the magnetite and wüstite layers. A sub-microscopic porosity in these layers has therefore been assumed, although the microscopic investigations show dense structures. It appears, however, that this assumption is unnecessary. According to the mechanism of reduction proposed here, the boundaries between wüstite and iron or gas are the only places where a gaseous reaction product has to be removed, and here the microstructure is really porous. The only exception is the enveloped Fe_3O_4 islands. The main part of the reduction, the breakdown of the oxygen lattice, takes place at these surfaces. The distances of diffusion velocities through these layers cover the reduction rates at fast as well as at slow reactions. When an ideal magnetite crystal is reduced by CO at 1000° C. (a very slow reaction (Fig. 5)) the reduction curve is linear until 60–70% oxygen has been removed and then becomes more parabolic, which indicates that only towards the end of the reduction does the diffusion govern the reaction velocity.

Small contents of H_2O and CO_2 in the reduction gas strongly retard the velocity of the wüstite reduction.[30] This illustrates the influence of the external gas velocity and gas diffusion on the reduction rate. High velocities give a good gas exchange and low partial pressures of H_2O and CO_2 at the reduction surface, which means favourable conditions for the slowest part of the reduction.

Formation of Pores in Reduction of Fe_2O_3 and Fe_3O_4

The formation of voids is an essential part of the diffusion process. In metallographic studies of so-called diffusion couples with different characteristic diffusion coefficients of the diffusing particles, a considerable porosity is formed in the part of the sample that loses mass through unequal diffusion and which decreases in size, causing the so-called Kirkendall effect.[61, 62, 63, 64] This may occur irrespective of the diffusion mechanism, and diffusion pores may be formed through great deviations from the equilibrium density of imperfections in the lattice, either as a super-concentration of vacancies or an abnormally low concentration of interstitial ions or atoms. Small voids, cracks, or inclusions may act as sinks for vacancies or sources for interstitial particles. Cavities are formed if a suitable supersaturation of the former or an abnormally low concentration of the latter is present.

If N_v/N_{vo} is equal to 2 or more (N_v = real vacancy concentration and N_{vo} = equilibrium vacancy concentration), vacancies are, for instance, condensed at spherical initiation voids with a volume corresponding to about 1000 vacant lattice points. In a perfect lattice without initiation points the tendency to formation of pores is considerably lower. Vacancies are condensed to form pores only when N_v/N_{vo} is about 100 or more.[64]

As described above, the swelling during reduction, and thereby the formation of oxide porosities, is closely associated with reduction steps which include diffusion in solid phase. Further, the expansion proceeds parabolically. Similar parabolic swelling has been observed during diffusion of a Cu–Ni sample at 1065° C.[65] The swelling in the direction of the diffusion gradient was then between one-third and one-half of the Kirkendall effect. One-third to one-half of the vacancy stream should thus have been transformed to pores.

It is probable that in reducing Fe_2O_3 to Fe_3O_4 the transformation from hexagonal to cubic oxygen lattice gives high possibilities for the generation of lattice imperfections. In the disturbed lattice the diffusion is accelerated and a large vacancy concentration or an abnormally low concentration of interstitial atoms should arise in parts which lose mass. The precipitation of voids is further facilitated in the defective lattice and these factors co-operate to give a considerable pore formation. If these pores from the beginning are so small that selective diffusion may proceed through them, a bursting action is obtained; CO or H_2 may pass through the pores, but CO_2 or H_2O formed may not come out. In the pores CO and H_2 thus have about the same absolute pressure as in the surrounding atmosphere, while the CO_2 and H_2O pressures rise above this value. Smith[66] has observed such a selective gas diffusion in connection with dezincing of brass by evacuation. The canals through which the zinc passed away, and which originated as diffusion pores, are sufficiently small to act as semi-permeable barriers for the gas, allowing small molecules to pass and holding back the big ones.

Disintegration of the structure causes an enlargement of the reaction surface and a shortening of the diffusion distances. The reaction and the diffusion are thus accelerated and the reduction has a more autocatalytical character.

In reducing, by CO, an ideal Fe_3O_4 crystal to wüstite—which may be crystallographically reversed without any change in the oxygen lattice[22]—diffusion in solid phase is slow and precipitation of holes is very improbable on account of the considerable supersaturation which would be necessary in the perfect lattice. The conditions for formation of pores during diffusion are thus absent. With a less ideally built Fe_3O_4, a higher tendency towards formation of pores may be expected. The boundary between wüstite and iron is also irregular in crystals which are less ideally built, and porosity has here been observed in the wüstite.

Reduction of crystals by H_2 is considerably faster than by CO, and the dense layers across which the diffusion in solid phase proceeds will be correspondingly thinner. A contributing cause to this may be that

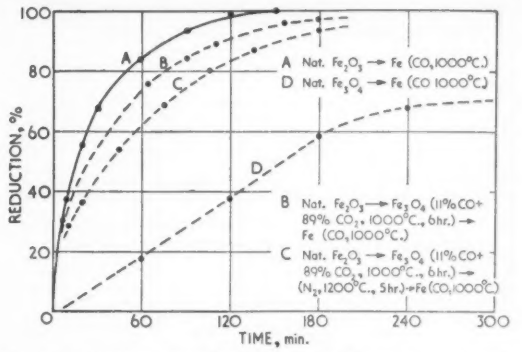


Fig. 45—Effect of annealing on reducibility of magnetite

H_2 enters small diffusion pores more easily than CO and afterwards bursts. With Fe_3O_4 as the original material, pore formation in the wüstite may be observed in reduction by H_2 in crystals where no pore formation is observed with reduction (other conditions being equal) by CO (Fig. 29).

To determine possible lattice defects in Fe_3O_4 reduced from Fe_2O_3, the width of X-ray diffraction lines from the reduced sample was compared with the line width from natural Fe_3O_4, by means of a spectrometer with a Geiger-Müller counter. There is always difficulty in proving lattice defects formed at high temperature owing to the equalizing influence of heat-treatment at the reaction temperature and during cooling. However, a small remaining disturbance was proved.

That annealing at reduction temperature influences the reducibility of the intermediate steps in Fe_3O_4 reduction may be seen from Fig. 45. Curve *a* shows oxygen percentage removed as a function of time in reducing Fe_2O_3 directly to Fe. Curves *b* and *c* represent the reactions of different intermediate annealings of Fe_3O_4 reduced from Fe_2O_3. It is probably not possible to anneal a sample so that the reaction curves correspond completely to a natural Fe_3O_4 single crystal. The intermediate Fe_3O_4 phase is always polycrystalline and contains elongated pores (Fig. 25). The porosity cannot be removed by annealing when the solid phase is present, but the curves appear to indicate that lattice defects, which disappear more or less during annealing, influence the reducibility of the oxides.

The elongated type of porosity, formed in this case in the oxide phase during the Fe_3O_4 stage, is not increased by further reduction to Fe of the annealed oxide (Fig. 37). Between the pores the structure of the iron is similar to that of iron from natural Fe_3O_4 (Fig. 17).

Reduction of Ores and Pellets

To determine whether the structures observed with pure crystals are formed also in the reduction of ores, a hematite from El Pao, Venezuela, and a magnetite from Stripa, Sweden, were reduced with CO at 1000° C. and microscopically examined. Both ores were crystalline, and 4-mm. cubes were reduced. The hematite contained 69·67% trivalent iron and the magnetite 71·22% total and 23·4% bivalent iron.

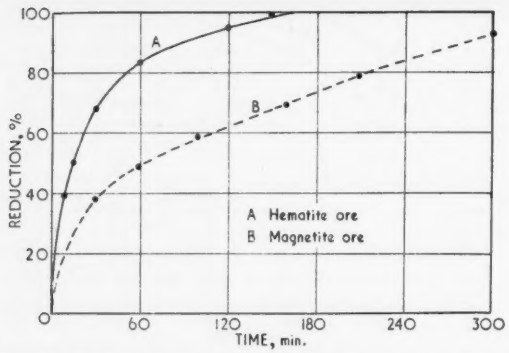


Fig. 46—Reduction of a hematite and a magnetite ore with CO at 1000° C.

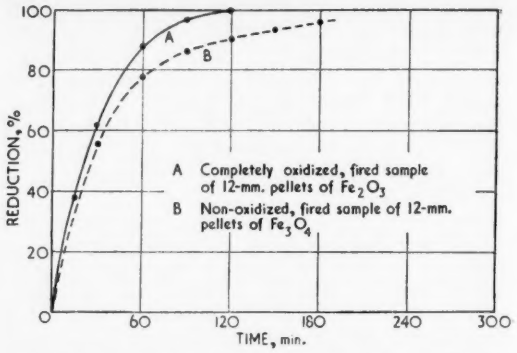

REDUCTION, %
TIME, min.
A Completely oxidized, fired sample of 12-mm. pellets of Fe_2O_3
B Non-oxidized, fired sample of 12-mm. pellets of Fe_3O_4

Fig. 47—Reduction of sinters with CO at 1000° C.

The difference in reducibility is very marked (Fig. 46), but not quite so much as with single crystals. The reducibility of the hematite ore corresponds to that of the hematite single crystal, but the magnetite ore is more easily reduced than the magnetite crystal. This difference is explained by the microstructures of the partially reduced ore samples (Figs. 38 and 39). The hematite ore and crystal have corresponding structures, but in the polycrystalline magnetite the crystal grains loosened from each other at the beginning of the reduction, and during the reaction an aggregate of small crystal grains was, in fact, reduced. The surface, and consequently the reducibility, of the wüstite is thus increased.

Well-burned, completely oxidized pellets prepared from pure magnetite concentrate without additions of binders have a very low strength during reduction in a sponge iron furnace of the Wiberg-Söderfors type, despite the high strength of the unreduced pellet. The rich pellets swell considerably during reduction, which causes difficulties in a shaft furnace. On the basis of the structural investigations of the reduction mechanism, the following hypothesis concerning this low strength was advanced by the author.[67] The bonds in burned pellets are of three main types: slag, magnetite, and hematite.[68] In a well-oxidized, pure, burned pellet there are only hematite bonds. These may be supposed to be broken down during the first stages of reduction principally in the wüstite stage, when a structure with very coarse pores is formed (Fig. 40). To obtain a pure pellet, *i.e.*, consisting only of iron oxides, and with a high strength, burning under less strongly oxidizing conditions is required. The pellet will then contain magnetite bonds which retain their strength during reduction. This means, however, a loss in reducibility. Good strength combined with high reducibility may be obtained by adding certain binders. This was recently confirmed by experiments made in collaboration with P. A. Ilmoni. Pellets from the same batch, burned in excess air and in purified N_2, were compared. The pellets were made from Malmberget A-10 concentrate containing 71·6% Fe. The pellets burned to hematite had high reducibility but low strength. The other pellets, which consisted of magnetite, but had a similar porosity to the first ones, had low reducibility and high strength. When suitable binders, principally calcium compounds, were added, the pellets were both easily reduced and strong.

Figure 47 compares the reducibilities of 12-mm. pellets of hematite obtained by oxidation of magnetite and of magnetite burned in a N_2 atmosphere.

In Table III the final swelling of reduced pellets is shown. An addition of only 0·5% CaO prevents the swelling of burned pellets and influences the reducibility very slightly. Microscopical investigations show that the hematite grains are bound together by a grey phase, probably a calcium ferrite, which evidently acts as an effective binder during reduction. The reduced pellet has a very dense structure.

SUMMARY

(1) The mechanism of reduction of natural hematite and magnetite single crystals by CO was studied at temperatures between 800° and 1100° C. Reduction curves confirm that there is a considerable difference in the reducibility of the two minerals. The faster reduction of Fe_2O_3 was shown by microscopic studies to be due to an earlier and more extensive formation of pores in the reaction products. When an Fe_3O_4 single crystal is reduced by pure CO at 1000° C. the wüstite layer is dense and all the porosity is formed only during reduction to the metal stage. When an Fe_2O_3 single crystal is reduced, a considerable porosity is obtained in the wüstite, which accelerates the continuing reduction to metallic iron. The magnetite layer in a partially reduced Fe_2O_3 crystal is dense. The phases formed were also identified by X-ray crystallographic means.

Table III

VOLUME CHANGES OF PELLETS IN REDUCTION BY CO AT 1000° C. AFTER DIFFERENT PRELIMINARY TREATMENTS

Specimens 1–4 burnt in excess air, 5 and 6 burnt in N_2

Specimen	Additions	Porosity before Reduction, %	Volume Change, %
1	None	24·1	+59
2	"	24·5	+60
3	0·5% CaO	26·3	− 8
4	"	26·4	− 5
5	None	28·6	− 5
6	"	26·8	− 3

(2) Fe_3O_4 crystals oxidized to Fe_2O_3 have reduction rates and structures similar to those of natural Fe_2O_3 crystals.

(3) Reduction of crystals, especially magnetite, by H_2 is faster than by CO at about 1000° C. The structures are, on the whole, similar with both reducing agents, but with H_2 the porosities are finer and the dense layers are thinner, and therefore the diffusion distances in solid phase are shorter. Pores are then formed also in wüstite from magnetite, and the difference in reduction rate of Fe_2O_3 and Fe_3O_4 is not as high as with CO.

(4) The temperature dependence of reduction by H_2 was studied from 450° to 1000° C. The difference in reactivity of Fe_2O_3 and Fe_3O_4 decreases at lower temperature, and at 450° C. has disappeared. A minimum reaction rate appears above 600° C. when both Fe_2O_3 and Fe_3O_4 are reduced. Phase studies under the microscope and with X-rays have shown that this reaction minimum is connected with the formation of continuous layers of wüstite at these temperatures. When wüstite does not appear, a cracked and finely porous iron is obtained and the magnetite surface is free for direct gas action.

(5) Changes in dimensions during reduction by CO were measured on Fe_2O_3 and Fe_3O_4 crystals. An Fe_2O_3 crystal swells considerably at first, but shrinks during the later stages. The final change is a swelling of 40–50% of the apparent volume. Fe_3O_4 crystals swell a little at first, but shrink later on, and the final result is a decrease in the apparent volume.

(6) The mechanism of reduction of Fe_2O_3 and Fe_3O_4 and the different tendencies to formation of pores during reduction were discussed in connection with the lattice relations of iron oxides.

(7) Reduction of two pure crystalline ores gave rise to structures very similar to those of the crystals. Results corresponding to those found in the crystals were also found in the reduction of pellets of magnetite or hematite. The poor strength of pure hematite ores and well-oxidized, rich pellets during reduction by CO appears to be due to the early and extensive formation of pores in the reduction products from Fe_2O_3, which is accompanied by a considerable swelling and a breaking up of the structure.

Acknowledgments

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The Length of Oil and Gas Flames

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SYNOPSIS

There is a need for a simple relationship between flame length and the quantities and velocities of air and fuel passing through the burner. Such relationships determining flame length are deduced from the theory of turbulent jets in forms suitable for application to oil and gas burners. The terms of the equations depend on measurements of flow, pressure, and density, together with an empirical coefficient.

An account is given of experimental work designed to verify this relationship for town gas and gas oil burning in open vertical flames. Good agreement between theory and practice was found for straight jets, and it was possible to extend the same form of equation to swirling jets.

The experimental values of the equation coefficient may not be directly applicable to works practice because of the difference in conditions, but suitable values are easily obtainable by experiment when required. For some purposes it is not essential to know the exact value of the coefficient. The relationship cannot be applied to very long lazy flames, nor to turbulent flames in very narrow combustion chambers.

Some data on the interference between two flames and impingement on a flat surface are given, showing the order of magnitude of these effects. 818

Introduction

The selection of appropriate burners and suitable operating conditions for steelworks furnaces depends largely on previous experience. But the duties demanded of furnaces vary so widely that there may, in fact, be little previous experience to act as a guide ; this is particularly true when liquid fuel, which has only recently been widely used in the steel industry, is involved. Even when appropriate data exist, they are not necessarily available to the works engineer, who has to decide between several alternative methods of firing a furnace. Consequently there is a large element of guesswork involved, and much expensive trial and error may be necessary before suitable conditions are found.

When selecting burners for a furnace or changing the operating conditions in an existing installation, although heat transfer must be the main criterion, it is also important to know the flame length to be expected, so as to ensure that the required quantity of fuel can be burnt in the space available.

The mechanism of heat transfer from luminous

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flames is being studied in detail by the Flame Radiation Committee,[1] but it was considered desirable to attempt to find a simple relationship between flame length and flow conditions at the burner which will assist the works engineer in the solution of problems involving the selection and operation of gas and oil burners, as a first step in the project on flame control.

Steelworks furnaces are nearly always fired with gaseous fuel. When coal is the primary fuel, it is usually converted into gas (producer, coke-oven, or blast-furnace), before it is used. Oil must be evaporated before it will burn ; this is achieved by injecting the oil into the furnace as a fine spray with a large surface area for the absorption of heat. Provided that the spray is fine enough, evaporation proceeds at a greater rate than combustion, so that again the problem is that of mixing a gaseous fuel with its combustion air.

On occasions, the air and fuel gases may be premixed before they are introduced into the combustion space, but it is more usual for mixing to proceed simultaneously with combustion in the combustion chamber itself. Since at flame temperatures the chemical reactions involved take place at a great rate once the fuel and air have been brought into intimate contact, the rate of combustion and the dimensions of the flame will probably depend very largely on the rate of mixing. This was suggested by the results of some experiments on oil burners in the open-hearth furnace.[2] The purpose of the present paper is to derive, from the theory of mixing in jets, relationships of a convenient form to describe flame behaviour and to give an account of experiments designed to confirm them. The practical applications of the results obtained are discussed separately.[3]

List of Symbols

A	Area of nozzle, sq. ft.
a, b	Proportionality constants (dimensionless)
C	Discharge coefficient (dimensionless)
G	Momentum flux, lb. ft./sec./sec.
K	Empirical coefficient (dimensionless)
k	Empirical coefficient, $\sqrt{}$ (cu. ft./lb.)
L	Visible flame length, ft.
M	Total mass flowrate across a plane section perpendicular to the axis at a distance z from the nozzle, lb./sec.
M'	Total mass flow at visible tip of flame, lb./sec.
m	Mass flow of jet fluid lb./sec.
q	Volumetric discharge rate of jet fluid, cu. ft./sec.
r	Ratio of primary air to fuel, lb./lb.
R	Mass ratio of jet fluid to fuel, lb./lb.
s	Average density of flame gases, lb./cu. ft.
s'	Density of jet fluid, lb./cu. ft.
u	Adiabatic discharge velocity of jet fluid, ft./sec.
w	Mass flowrate of fuel, lb./sec.
z	Distance from nozzle to any point on the flame axis, ft.

DERIVATION OF FLAME-LENGTH EQUATIONS

Theory of Turbulent Jets

The behaviour of a free turbulent jet of air mixing with the atmosphere has been analysed by applying the concept of momentum transfer,[4] and the basic equation obtained is :

$$\frac{M}{z} = a\sqrt{(Gs)} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (1)$$

This equation applies only to a turbulent jet, and the nature of turbulent flow must be clearly understood. A turbulent stream of gas is not homogeneous in structure but may be considered as a succession of separate parcels of gas. These parcels have random motions in all directions superimposed on their steady motion in the direction of flow. These random motions produce a jostling action, as a result of which neighbouring parcels constantly exchange gases, so that mixing takes place. Transfer of momentum takes place simultaneously, and when a parcel of gas at the boundary of the jet collides with a stationary parcel in the surrounding air the latter parcel acquires momentum and is entrained into the jet.

Both entrainment of air into the jet and mixing in the jet thus depend on the same fundamental mechanism, and a relationship which describes one process should therefore apply also to the other. The above equation was obtained to describe the process of entrainment ; it will be assumed that it also applies to the process of mixing.

The term ' turbulent flow ' implies random motion, but it does not necessarily imply any sort of rotational flow. The term is often applied to flow in which the stream is rotating about its axis of flow. In this paper confusion will be avoided by using the term ' turbulent ' in its strict sense and by using the adjective ' swirling ' to describe rotating flow.

A flame differs from the above simple system because it results from the mixing of two fluids of different density, both of which may be turbulent, under conditions in which natural convection due to difference in temperature will also promote mixing. The basic equation can be applied to flames with sufficient accuracy for industrial purposes, provided that the mixing action of one of the streams is large compared with the remaining effects. Such a flame may be termed a ' jet flame ' to distinguish it from other types of flame, such as ' diffusion ' flames.

Definitions

Jet Fluid—That part of the total flow which is the source of the momentum flux controlling the mixing process.

Primary Air—Air flowing through the same nozzles as the fuel, *e.g.*, air mixed with gas or atomizing air in an oil burner.

Secondary Air—Air admitted to the flame through an orifice separate from that used for the fuel with sufficient momentum to make an appreciable contribution to the mixing process.

Tertiary Air—Air entrained by the jet and having insufficient momentum to make an appreciable contribution to the mixing process.

The Flame-Length Equation

Equation (1) may be written in the form

$$L = \frac{M'}{a\sqrt{(Gs)}}.$$

Assuming that M' is proportional to the fuel flow w,

$$L = \frac{bw}{a\sqrt{(Gs)}} = K\frac{w}{\sqrt{(Gs)}}.$$

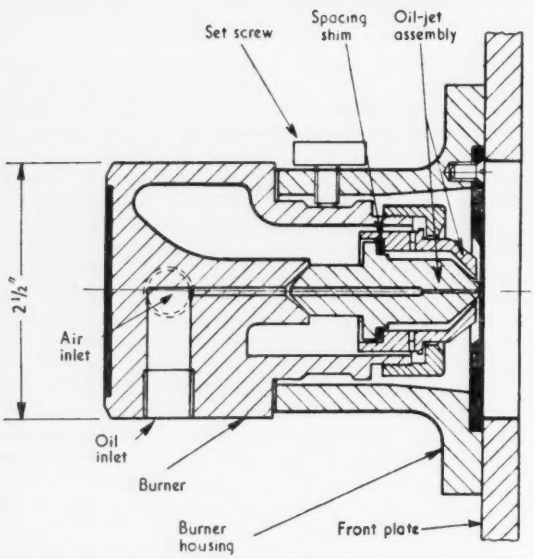

Fig. 1—General arrangement of single-stage outside-mixing atomizer

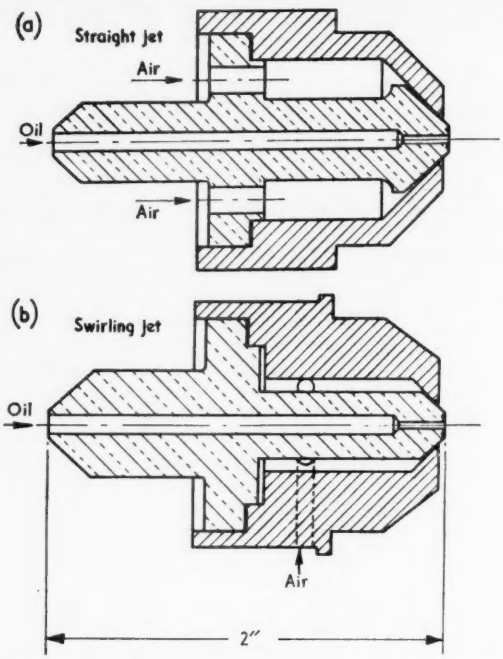

Fig. 2—Details of straight and swirling jets for single-stage atomizer

Because of the difficulty of assigning a value to s, this has been combined with the empirical coefficient, and the experimental data have been correlated in the form

$$L = k \frac{w}{\sqrt{G}}.$$

In this form k is, of course, no longer dimensionless.

The momentum flux G has been evaluated from the measured flowrate of the jet fluid, and a velocity has been calculated from the pressure drop on the assumption of frictionless adiabatic expansion. This introduces some error, since no real nozzle can be wholly frictionless, but friction losses in normal nozzles with smooth surfaces are small, so that the error from this cause should not be excessive.

EXPERIMENTAL VERIFICATION

Experimental work has been carried out with the objectives of

(i) Verifying the theoretical relationships deduced for simple jets and determining the empirical coefficients
(ii) Ascertaining whether equations of the same form could be applied to swirling jets.
(iii) Measuring the effects of interference between flames and impingement of a flame on a flat surface.

This work has been carried out under laboratory conditions using gas oil and town gas as fuel. The values of the empirical coefficients are therefore not necessarily directly applicable to actual furnace practice, but it is believed that the principles established by the experiments will be equally applicable to works conditions.

Apparatus

Gas Burners—These were simple pieces of tube, of length not less than ten diameters, which were made up as required.

Oil Burners—Three different types were used :

(*a*) Single-stage outside-mixing burner (swirling and straight-flow forms—*see* Figs. 1 and 2) : This is a low-capacity medium-pressure air atomizer suitable for burning gas oil. Oil is delivered at low head to the centre of a ' vortex cup ' surrounded by an annular air nozzle. This annular jet sets up a ring vortex in the vortex cup which spreads the oil in a film over the face of the cup ; the oil is then sheared off the edge of the cup by the air blast as a fine spray. The burner in made in two forms (*see* Fig. 2) : one has straight-through air flow, and in the other air is admitted to the nozzle chamber through tangential holes to set up

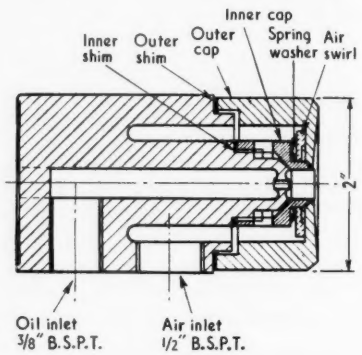

Fig. 3—Two-stage outside-mixing atomizer; alternative concentric-tube oil and air inlets at back of burner may be provided

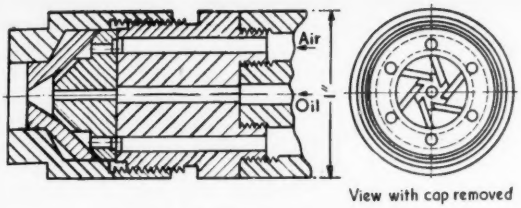

Fig. 4—Inside-mixing burner

a high degree of swirl. Swirling results in a broader and more stable flame.

(*b*) Two-stage outside-mixing burner (*see* Fig. 3) : This is a two-stage medium-pressure atomizer suitable for burning heavy oils. The first stage consists of a vortex cup and annular air nozzle similar to the single-stage burner. Beyond this is a second annular nozzle through which flows a further quantity of atomizing air. This air is normally given a high swirl, although in these experiments no swirl was used, and the relative proportions of the two air streams determine the shape of the flame. Both annular nozzles are adjustable by using spacing shims, giving an easily adaptable burner.

(*c*) Inside-mixing burner (*see* Fig. 4) : This is also a small medium-pressure air atomizing burner, of rather different design from burners *a* and *b*. Atomization takes place inside a conical nozzle, to which oil is admitted through a central tube, and atomizing air enters through six slots in the surface of the inner conical piece. These slots are tangential to the small-end circle of the conical piece, so that swirling air flow is obtained.

The operating characteristics for the three burners may be summarized as follows :

	Outside-Mixing Single-Stage	Outside-Mixing Two-Stage	Inside-Mixing
Maximum oil rate, gal./hr.	3	10	2·5
Maximum atomizing air pressure, lb./sq. in.	4	10	10
Atomizing air as a fraction of the combustion air, %	4	4	4
Oil pressure, lb./sq. in.	Negligible		10

These three burners represent two main classes of burner. In the outside-mixing burners, the air blast expands completely to atmospheric pressure before striking the oil ; in the inside-mixing burner the atomizing air and oil are mixed before the expansion is completed through the final orifice. This tends to have an important effect on the flow characteristics of the burners. In an outside-mixing type, the pressure in the atomizing zone is atmospheric, or possibly slightly less than atmospheric, so that a low oil-pressure is adequate and there is no mutual inference between the two flows.

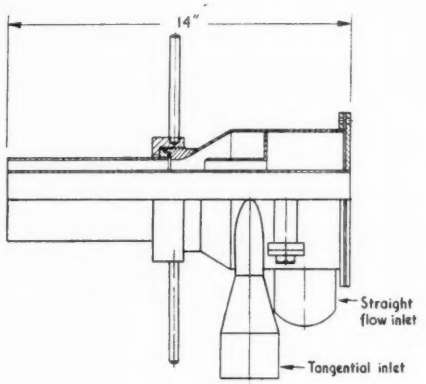

Fig. 5—Secondary-air nozzle

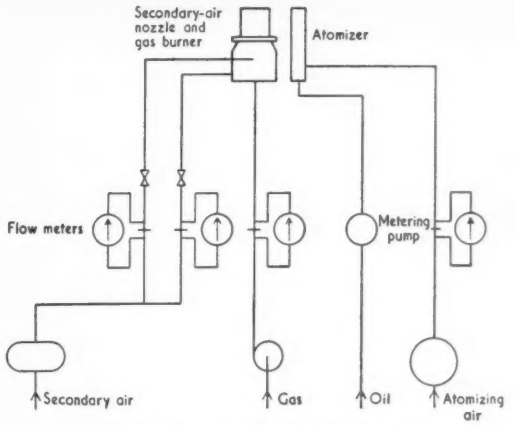

Fig. 6—Flow diagram of apparatus

In an inside-mixing burner the pressure in the mixing zone may be nearly as great as the initial atomizing air pressure, so that the oil must be delivered at not less than this pressure, which will vary with the quantity of air flowing. As the mixture expands through the final orifice, the air accelerates to a higher velocity than the oil and so exerts drag forces on it. These forces react on the air and effectively constitute an added resistance to flow, which varies with the quantity of oil. There is therefore considerable mutual interference between the two flows, which affects the ease of control. An outside-mixing burner may act as its own metering orifice and be controlled by pressure only, under all conditions, but this is not easily achieved with an inside-mixing burner. Some designs may even be unstable under some flow conditions, although this is not true for the burner under consideration in any part of its normal working range.

Secondary Air Nozzle—The secondary air nozzle, shown in Fig. 5, consists of an annular nozzle surrounding an inner tube large enough to accommodate most of the burners used in this work. Three sizes of annulus were available by using interchangeable external nozzles. Air was admitted to the nozzle through two alternative entry chambers. One had straightening vanes to ensure a non-swirling air flow, and the other had a tangential air inlet to provide maximum swirl.

Ancillary Apparatus—The oil was supplied and metered by means of a metering pump. Low-pressure fuel gas was delivered by a booster fan to overcome the resistance of the rotameter used for its measurement, and high-pressure gas was delivered by a rotary compressor with a positive displacement

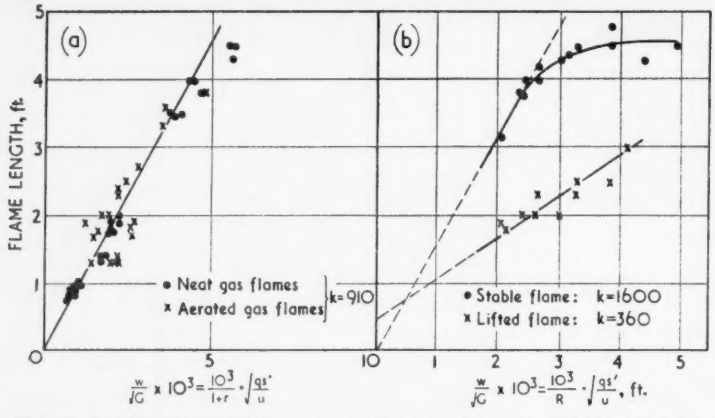

Fig. 7—Gas flames (*a*) without secondary air, (*b*) controlled by straight secondary air

meter in the suction. Positive blowers were used for the atomizing and secondary air, which were metered by a rotameter and orifice meters, respectively. Pressures were indicated by Bourdon, mercury, and water gauges as appropriate. A flow diagram of the apparatus is given in Fig. 6.

Experimental Procedure and Results

The burners were set up vertically in free air in the centre tube of the secondary air nozzle. A gas pilot-flame was placed near the burner nozzle and was kept continuously alight to maintain the main flame if blow-off conditions were reached. In general, however, the data given refer to stable flames. No sort of refractory surround was used to assist flame stability.

A scale was set up vertically at the side of the flame and the height of the flame was estimated visually. This could be done with reasonable accuracy with luminous flames, but it was rather uncertain with non-luminous flames. Gas oil was used as the liquid fuel. Because of its greater volatility compared with heavy fuel oil, the evaporation rate should be great enough to ensure that mixing is the main factor controlling the flame, although the intensity of radiation in the experimental flame was less than that usual in a furnace.

The tests carried out are described below and the experimental results are summarized in Table I. All data are expressed in self-consistent pound - feet - second units and are plotted in Figs. 7–13. Values of the coefficient k have been found from the slopes of these curves.

Tests 1 *and* 2: *Straight Jets of Fuel/Gas and Gas/Air Mixtures* (*Fig.* 7*a*)—In test 1 the jet fluid was the fuel itself, and the velocities ranged from 100 to 1340 ft./sec. ; the latter figure is approaching the sonic velocity for coal gas. In test 2, velocities were 50–115 ft./sec. and the amount of primary air mixed with the gas varied from 12 to 55% of the theoretical combustion air. The results of both tests can be represented by the equation

$$L = 910 \frac{w}{\sqrt{G}}.$$

A few of the longer flames were shorter than the equation indicates, showing that convective mixing was beginning to have a noticeable effect. When this occurs, the equation for jet flames will no longer apply.

Test 3 : *Gas Flames Controlled by Straight Secondary Air* (*Fig.* 7*b*)—In practice the jet fluid would comprise the whole of the combustion air in this type of

Table I

SUMMARY OF TESTS 1–9

Test No.	Burner Class	Jet Fluid	Fuel	Range of u, ft./sec.	Range of R	Primary or Secondary air, % of theoretical	Equation Coefficient	Figure No.
1	1	Non-swirling fuel gas	Town gas	100–1340	1	...	910	7*a*
2	2	Non-swirling primary air-fuel gas mixture	,,	50–115	2·3–7·2	12–55	910	7*a*
3	5	Non-swirling secondary air	,,	17–72	5·8–17·4	53–160	1600 (normal flame) 360 (lifted flame)	7*b*
4	4	Non-swirling primary air	Gas oil	480–960	0·6–2·4	4·3–17	1380	8
5	4	Non-swirling primary air	,,	500–700	0·7–1·3	5–9·3	1560	8
6	5	Swirling secondary air	Town gas	52–180	5·2–17·4	47–160	1640 (long flames) 760 (short flames)	9
7	1 and 5 combined	Swirling secondary air and non-swirling fuel gas	,,	54–220 air 100–200 gas	5·2–17·4	47–160	...	10
8	4	Swirling primary air	Gas oil	610–820	0·5–1·5	3·6–11	1360	11
9	4	Swirling primary air	,,	810–990	0·25–0·75	1·8–5·4	1100	11

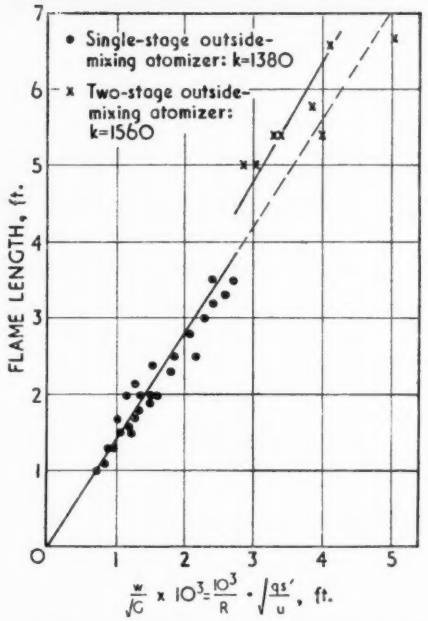
• Single-stage outside-mixing atomizer: k=1380
x Two-stage outside-mixing atomizer: k=1560
FLAME LENGTH, ft.
$\frac{w}{\sqrt{G}} \times 10^3 = \frac{10^3}{R} \cdot \sqrt{\frac{qs'}{u}}$, ft.

Fig. 8—Oil flames controlled by straight primary air

burner, but in these experiments a wide variation of the air flow was used to increase the range of the experimental data.

For this test the annular secondary-air nozzle was used. The gas velocity was 4 ft./sec., which by itself gave a ' lazy ' type of flame. Applying secondary air at first made little difference to the flame ; then the flame became shorter and more turbulent in character, and finally a ' lifted ' flame resulted. The flame separated from the tip of the nozzle, leaving a gap of about 2 in., and became shorter and non-luminous.

The results obtained are plotted in Fig. 7*b*. They fall into two sets, corresponding to normal and lifted flames, for which the respective values of k are 1600 and 360 ; the curve for the lifted flame does not go through the origin. The two sets of data overlap

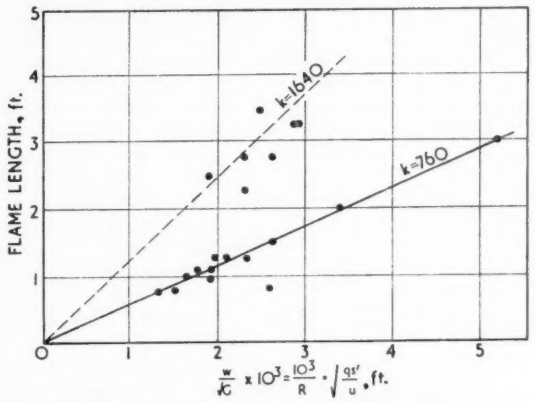

Fig. 9—Gas flames controlled by swirling secondary air

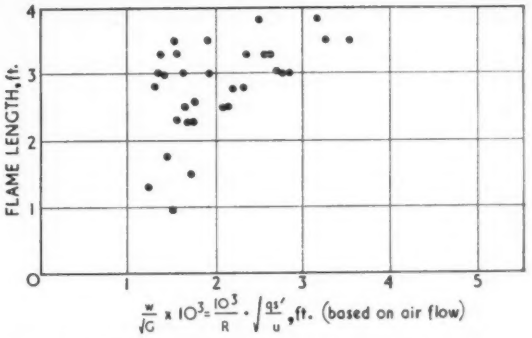

Fig. 10—Flames obtained with high-velocity gas and swirling secondary air

because the occurrence of blow-off seems to depend on both the air velocity and the size of the nozzle.

The difference between the two sets of data probably arises from premixing of fuel and air occurring in the gap between the nozzle and the base of the flame, so that it is doubtful if the value of k has been correctly calculated for this case. The value of k for the lifted flame should therefore not be compared with other values. In any case, the lifted flame is not of industrial importance as it represents the first stage of flame blow-off.

The quantity of secondary air used in this test varied from 53 to 160% of the theoretical combustion air, and the velocity varied from 17 to 72 ft./sec.

Tests 4 *and* 5 : *Oil Flames Controlled by Straight Primary Air* (*Fig.* 8)—Data for outside-mixing oil burners are plotted in Fig. 8. The results for the single-stage burners agree with the equation

$$L = 1380 \frac{w}{\sqrt{G}} .$$

The data for the two-stage burner are inadequate but appear to fall on a different line of slope 1560. This difference may be due only to experimental error, but it could also be a real effect due to the difference in design between the burners. The primary air varied from 4 to 17% of the theoretical combustion air and the air velocity varied from 480 to 960 ft./sec.

Test 6 : *Gas Flames Controlled by Swirling Air* (*Fig.* 9)—The burner arrangement was the same as for test 3, but the tangential secondary-air inlet was used. The air was varied over a wider range than is usual in this type of burner, and the gas velocity was 4 ft./sec., as before. With low quantities of secondary air the flame was not much affected, although spiral flow lines were visible in it. On increasing the secondary air there was a rapid reduction in length until a compact pear-shaped flame was obtained. This flame was completely stable and showed no signs of flame lift.

The test results are shown in Fig. 9. The data fall into two groups, corresponding to long and short flames, for which the values of k are 1640 and 760, respectively. The secondary air varied from 47 to 158% of the theoretical combustion air, the transition from ' long ' to ' short ' flame conditions occurring at

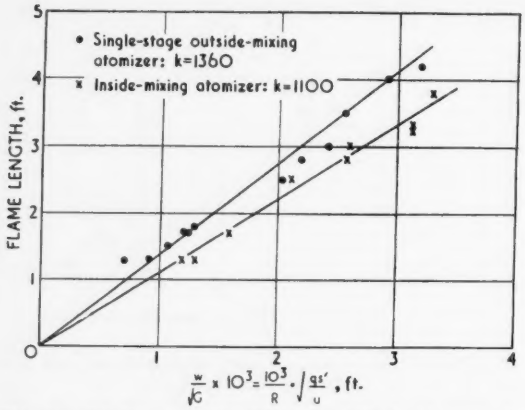
• Single-stage outside-mixing atomizer: k=1360
x Inside-mixing atomizer: k=1100
FLAME LENGTH, ft.
$\frac{w}{\sqrt{G}} \times 10^3 = \frac{10^3}{R} \cdot \sqrt{\frac{qt'}{u}}$, ft.

Fig. 11—Oil flames controlled by swirling primary air

about 80%. The air velocity varied from 52 to 180 ft./sec.

Associated with the change to short-flame conditions was the development of a strong back-flow in the central core of the swirling jet. By feeding back burning gas to the root of the flame, the stability of the flame was greatly increased.

Test 7 : *Flames Controlled by the Combined Action of High-Velocity Gas Jets and Swirling Secondary Air* (*Fig.* 10)—In this test air velocities ranged from 54 to 220 ft./sec. and gas velocities were 100, 150, and 200 ft./sec. In Fig. 10 flame length is plotted against $w/\sqrt{G}$ for the air flow, as in the previous tests with secondary air. The results show considerable irregularity and no definite correlations are apparent.

The flames themselves were irregular in appearance with floppy tips, and the compact flames given by the combination of swirling secondary air and low-pressure gas were not obtained. A further qualitative test with an air-atomized oil burner was also made. The burner was set up in the secondary-air nozzle and operated with both straight and swirling secondary air, equal quantities being used in the two cases. The straight air flow gave a clean sharply-defined flame ; the swirling air flow gave a smoky flame with an irregular tip.

Tests 8 *and* 9 : *Oil Flames Controlled by Swirling Primary Air* (*Fig.* 11)—These tests were carried out with an outside-mixing and an inside-mixing oil burner. The results gave the following values of k :

Outside-mixing burner	1360
Inside-mixing burner	1100

The difference between the two values of k can reasonably be assigned to the difference in design between the burners.

The quantity of primary air varied from 1·8 to 11% of the theoretical combustion air, and the air velocities ranged from 610 to 99 ft./sec.

Test 10 : *Interference between Flames* (*Fig.* 12)—Some data on the interference between two straight gas jets are plotted in Fig. 12. The separation of the burner nozzles is in terms of the greatest flame diameter.

Test 11 : *Impingement of a Flame on a Flat Surface* (*Fig.* 13)—The changes in flame length which result from impingement on a flat surface are plotted in Fig. 13. All lengths are expressed in terms of the ' free ' flame length.

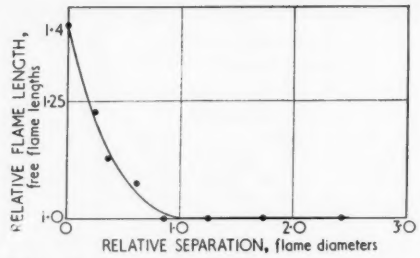

Fig. 12—Interference between two flames

DISCUSSION

Application of the Flame-Length Equation

The results obtained strictly apply only to open flames, but Thring and Newby[5] have shown that results based on free jets can be applied to enclosed flames subject to certain corrections. Limitation of excess air results in elongation of the flame ; this elongation is about 25% when working with 30% of excess air.

A jet expanding in a chamber tends to touch the walls at about three chamber diameters from the nozzle, and this alters the flow pattern so that free jet conditions no longer apply. The flame-length equation may therefore be applied to flames not exceeding three chamber diameters in length, but not to longer flames.

No work has been carried out on high-pressure burners operating above 10 lb./sq. in. It is probably safe to apply the equation to gas jets and outside-mixing oil burners, provided that care is taken to estimate the velocity correctly. If the nozzle is convergent only, the velocity cannot exceed the sonic velocity appropriate to the conditions in the nozzle throat, but with a well-designed convergent–divergent nozzle almost the full adiabatic velocity will be obtained.

It is also inadvisable to apply the equation to inside-mixing oil burners because of the difficulty of calculating the velocity correctly. The equation does in fact apply to one inside-mixing burner described in this paper, but the value of k obtained differed from that

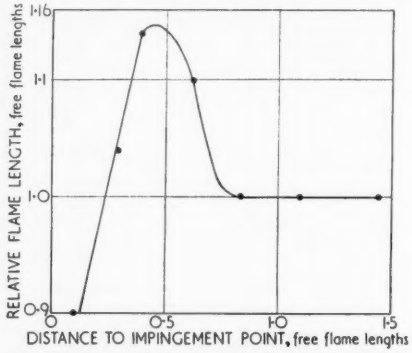

Fig. 13—Impingement on a flat surface

obtained with outside burners. This different value may well be characteristic of the particular burner used and should not be applied to a different design of burner without experimental evidence of its validity.

Significance of the Flame-Length Equation

To assess the significance of the equation it is convenient to change it to a different form. If m is the mass flowrate of the jet fluid and s' is its density as it leaves the nozzle,

$$L = K\frac{w}{\sqrt{(Gs)}} = K\frac{w}{\sqrt{(mus)}} = K\frac{w}{m}\sqrt{\frac{m}{us}} = \frac{K}{R}\sqrt{\left(\frac{q}{u}\cdot\frac{s'}{s}\right)}$$

This transformation has introduced no new constants and so the value of K is the same for both forms of the equation.

Burners can be classified according to the nature of the jet fluid. In the simple gas jet $m = w$, so that $R = 1$. In premixing gas burners where the jet fluid is a mixture of fuel and primary air, $R = 1 + r$. There is also a large group of industrial gas burners in which the gas enters at low velocity and therefore with negligible momentum, the jet fluid being all or part of the combustion air.

Oil burners using steam or air for atomization may also be included. In these burners the momentum contributed by the oil is negligible and the atomizing medium provides the jet fluid.

The significance of the various terms of the equation is discussed below.

Ratio of Jet Fluid to Fuel (R)—The flame length is sensitive to variations in R. This is particularly important in the case of oil burners, where it is often possible to vary R over a wide range without causing operating difficulties, thus giving a wide variation in flame length.

Effective Nozzle Area (q/u)—The ratio q/u is the effective area of the nozzle, so that this term represents the size of the burner. In most types of burner this is substantially constant, since it is affected only by variations in discharge coefficient, which is constant at the high Reynolds' numbers usual in practice. In the inside-mixing oil burner, however, the pressure required depends on the quantity of oil as well as on the quantity of atomizing fluid, so that in this case the ratio q/u as calculated in the paper will also depend on the ratio of atomizing fluid to oil.

It follows that if in a given burner both the jet fluid and fuel are varied in the same proportion, so that R is unchanged, the flame length will remain constant, apart from a small effect due to change in the flame density s as a result of a change in flame temperature. Any such temperature change will be relatively small, and will have the effect of lengthening the flame as the fuel rate increases.

Since for constant pressure the capacity of a burner will be proportional to its effective area, it follows that the flame length will be proportional to the square root of its capacity. If a given input is divided among a number of small burners, the flame length will vary inversely as the square root of the number of burners.

Density Ratio s'/s—This term shows that preheating the jet fluid will reduce the flame length by reducing s'. In oil burners, for equal values of R, the use of steam for atomizing should give a shorter flame than air, because the density of steam is about 0·55 that of air.

Effects that tend to increase the flame temperature (such as air preheat or high-temperature surroundings) will decrease the flame density s and so lengthen the flame. These effects will not be large, as the variation in flame temperature normally obtained has only a small relative effect on the absolute temperature.

Coefficient k—The coefficient k must be determined experimentally. It is derived from components found in the basic equation and an assumed proportionality between the fuel flow and the total flow at the tip of the flame. It is therefore probable that the value of k depends on both the type of fuel and the design of the burner. It will probably also be greater under furnace conditions than in the open air, for in the furnace there is less air available and mixing will be slowed down towards the tip of the flame, thus increasing the length.

Comparison between Non-Swirling and Swirling Flames

For reasons mentioned above, the lifted straight flame must be considered as abnormal, so that the values of k to be compared are as follows :

Gas Flames Controlled by Secondary Air

Straight Flow	Swirling Flow
1600	Low values of R: 1640 High values of R: 760

Oil Flames Controlled by Primary Air (Low Values of R only)

Straight Flow	Swirling Flow
1380, 1560	1100, 1360

It is apparent that at low values of R the use of swirl makes little difference to the value of k. At high values of R there was a marked reduction in the value of k for gas flames. Also, the back-flow in the swirling burner, continuously piloting the flame, gave great stability. In these experiments the straight flames were on the verge of blow-off at an air velocity of 70 ft./sec., but the swirling flame was completely stable at 180 ft./sec. A swirling burner can therefore, for a given capacity, work with a lower value of q/u, giving a further reduction in flame length. The diameter of the flame, however, is relatively great and the forward component of velocity is low, so that the swirling burner would not be suitable for an application requiring a narrow flame of high forward velocity.

Combination of High-Velocity Gas Jets and Swirling Secondary Air

In test 7 (Fig. 10) swirling secondary air was combined with high-velocity gas jets, the gas velocity being high enough to produce a turbulent flame by itself.

This is the only example in this paper of flames produced by the combined action of two turbulent streams. It is of great industrial importance because gas burners controlled by the combustion air often have swirling air streams, and there is a growing practice to fit oil atomizers to these burners to provide an alternative fuel supply. These atomizers nearly always use steam or compressed air as the atomizing medium and give a high-velocity narrow-angle jet.

The results obtained under these conditions are most irregular, as shown by Fig. 10. The flames themselves were ragged in appearance with floppy tips, and the compact globular flames given by similar air flow and low-pressure gas were not obtained. A similar result was obtained with the air-atomizing oil burner.

The strongly swirling secondary-air stream tends to expand radially and produces a reduction in pressure in the centre of the jet, causing a strong back-flow towards the burner. If the fuel velocity is low this back-flow deflects the fuel outwards into the main jet and rapid mixing results. If the fuel velocity is high, the fuel overcomes the back-flow and flows down the central core of the air jet, where the air supply is limited by the tendency of the swirling air stream to expand outwards. Consequently, mixing is retarded.

From this it can be concluded that the combination of swirling secondary air and a high-velocity narrow-angle fuel jet should be avoided. The fuel jet should either have a low velocity or be in the form of a hollow wide-angle cone, so that the fuel enters the main air stream at once.

Effects of Interference with Straight Jets

When a flame that is entraining air from its surroundings comes into contact with another flame or a solid body, the flow of air across part of the flame surfaces is obstructed. This loss of entraining surface must be made good elsewhere and a change in the size and shape of the flame is to be expected.

The effects of interference between two free-gas jets is shown in Fig. 12 (test 10). There is no apparent increase in length until the distance between the axes is less than 0·75 flame diameters, measured at the greatest diameter of the flame. There is then a progressive increase along an approximately parabolic curve, which, in the extreme case of complete coalition of the nozzles into one of twice the area, passes through a relative flame length of $\sqrt{2}$, as is required by the form of the mixing equation. This is the case where a flame has interference on one side only. In a row of jets each flame except the end ones has interference on two sides, so that the elongation would be greater. Elongation does not occur until the flames are in close contact near their mid-points, so that all the air required for combustion must be entrained in the first half of the flame. The effect of close bunching of multiple flames could probably be roughly estimated from the loss of free surface for air entrainment in the first half of the flame.

The effect of impingement on a flat surface is probably due also to changes in exposed surface (Fig. 13). When the nozzle is close to the surface, the flame is so much flattened that its area is increased and it becomes shorter. When the impingement point is near the middle of the flame, it is not so much flattened, and access of air to the underside is obstructed, lengthening the flame.

CONCLUSIONS

(1) Equations defining the length of non-swirling jet flames have been derived from the theory of turbulent jets and confirmed by experiment over a wide range of conditions for gas oil and town gas.

(2) Experiment has shown that similar equations can be applied to swirling jets.

(3) The equations are not valid for :

(i) Lazy (' diffusion ') flames

(ii) Flames where mixing is the result of the combined effect of more than one stream

(iii) Flames in narrow combustion chambers of diameter less than one-third of the free flame length.

(4) Values of the empirical coefficient in the equations as found in the laboratory may be expected to differ from those applicable to industrial conditions, but the relative effects of changing operating variables may be summarized as follows :

Variable Changed	Effect on Flame Length
Jet-fluid/fuel ratio increased	Decrease
Jet fluid and fuel changed in same proportion	No change
Velocity increased by change of burner size	Decrease
Jet-fluid temperature increased	Decrease
Flame temperature increased	Increase

(5) With low values (up to 8 lb./lb.) of the jet-fluid/fuel ratio, there is little difference between swirling and straight jets, but when the jet fluid exceeds about 80% of the theoretical combustion air, swirl causes a large reduction of flame length.

(6) At higher air/fuel ratios, (greater than 8 lb./lb.) the swirling air stream has a pronounced back-flow, which stabilizes the flame at velocities that would cause blow-off with non-swirling air.

(7) To obtain the maximum effect of swirling secondary air, the fuel jet should be such that the fuel is immediately carried into the main air stream. A high-velocity narrow-angle fuel jet flows down the central core and poor mixing results.

(8) Lifted flames (the first stage of blow-off) differ in characteristics from normal non-swirling jet flames, possibly because of premixing between the burner mouth and the root of the flame. Further data on blow-off conditions are required to increase the usefulness of the equation.

(9) For otherwise comparable conditions, the effects of burner design were not large.

(10) Interference between flames leads to an elongation as a result of loss of air-entraining surface, but the effect does not begin until the separation of the flame axes is less than 0·75 flame diameters.

(11) Impingement at an oblique angle on a flat surface results in an elongation unless the impingement point is within 0·25 flame lengths of the nozzle, in which case there is a reduction in length.

Acknowledgments

Figures 1–4 are reproduced by courtesy of the manufacturers of the burners.

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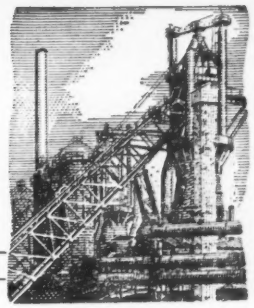
IRON AND STEELWORKS ENGINEERING

THE IRON AND STEEL ENGINEERS GROUP



The Ironmaking Plant at John Summers and Sons (Shotton Works)

By J. F. R. Jones, A.M.I.Mech.E., and A. Walker

SYNOPSIS

The paper describes the composition and operation of the ironmaking plant at Shotton, as it is at present, and of the completed plant, when the present development programme is finished.

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THE Shotton ironworks has been designed as a two-furnace plant, complete with the necessary coking capacity and all auxiliary services. The general layout is shown in Fig. 1. At present one blast-furnace is in operation, together with half the ultimate coking plant ; the second furnace, the remainder of the coking plant, and the sinter plant are at present under construction. The first furnace was blown in at the end of February, 1953. This paper will be confined to a general description of the plant, and it is hoped to augment this information at the meeting by records of actual operating experience.

The designed output of each furnace is about 7000 tons of iron per week on a prepared burden consisting of about 75% imported ore and 25% home ore, *i.e.*, an iron content in the burden of about 45%. It has, however, always been intended to operate on as near 100% imported ore as possible, so as to produce a grade of iron that is economically suitable for making low-carbon low-impurity steel which can be used for continuous strip rolling.

All imported ore reaches the works *via* Birkenhead Docks. A suitable ore dock with two modern unloaders has been built at Bidston, together with railway-siding and marshalling accommodation, and a fleet of 50-tons nominal capacity hopper bottom-bogie wagons brings the ore direct from Birkenhead to the works sidings, a rail haul of about 12 miles. Home ore arrives in various types of wagon, usually from Northants.

ORE HANDLING

The ore handling plant for imported ore consists of a line of 42 hoppers, each with a capacity of about 25 tons. A rail track runs along the top of these hoppers, so that a train of hopper wagons can be positioned over them and discharged in a very short time.

An apron feeder car is provided to empty the incoming material from the unloading hoppers ; this consists of a manganese-steel apron feeder mounted on a self-propelled carriage. The apron-feeder car is arranged to run on a track supported by the structure of the conveyor which receives the material fed out of the unloading hoppers.

To enable the rate of delivery of material to the crushing plant to be regulated, the apron feeder is fitted with a variable-speed drive having a 3 to 1 speed range, the normal speed of the feeder being 15–40 ft./min. The feeder is designed to handle a maximum capacity of 750 tons of iron ore per hour at 135 lb./cu. ft.

The hydraulic operating gear for the hopper gates is mounted on the feeder carriage. This gear comprises two hydraulic rams which, when positioned underneath the lever arms, open and close the gates under control.

The feeder car is positioned under one bunker and, when this bunker is empty, it passes to the next, and so on. The limit of travel of the car between each bunker is controlled by electrical limit switches, and the gate-operating gear is suitably interlocked.

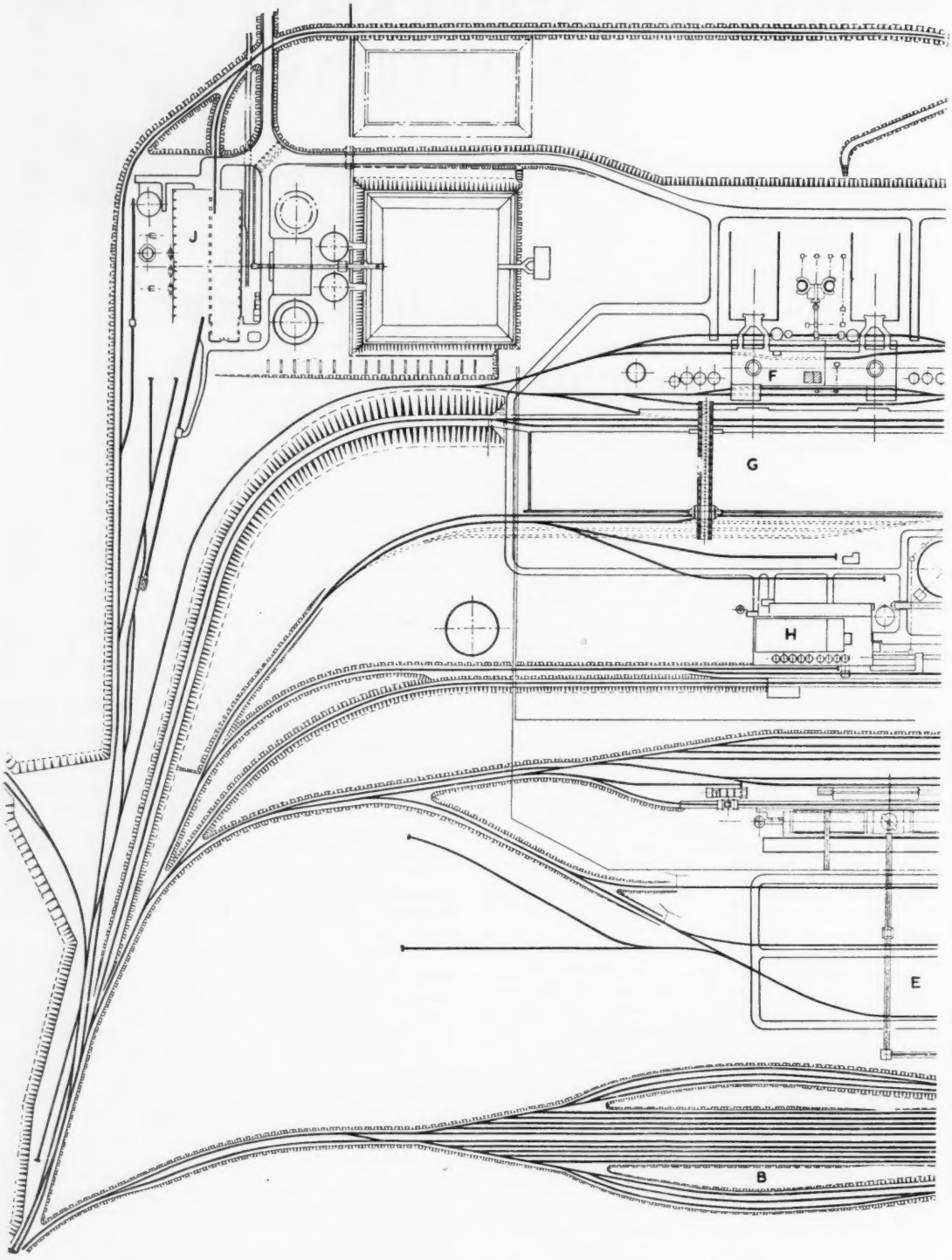
In practice it was found to be difficult to unload sticky home ore by this method, and so a side tippler discharging on to a drop-bar feeder has been installed at one end of the line of bunkers for handling this difficult material.

CRUSHING PLANT

Ore is delivered from the unloading conveyors into a receiving hopper for feeding the drop-bar feeder ;

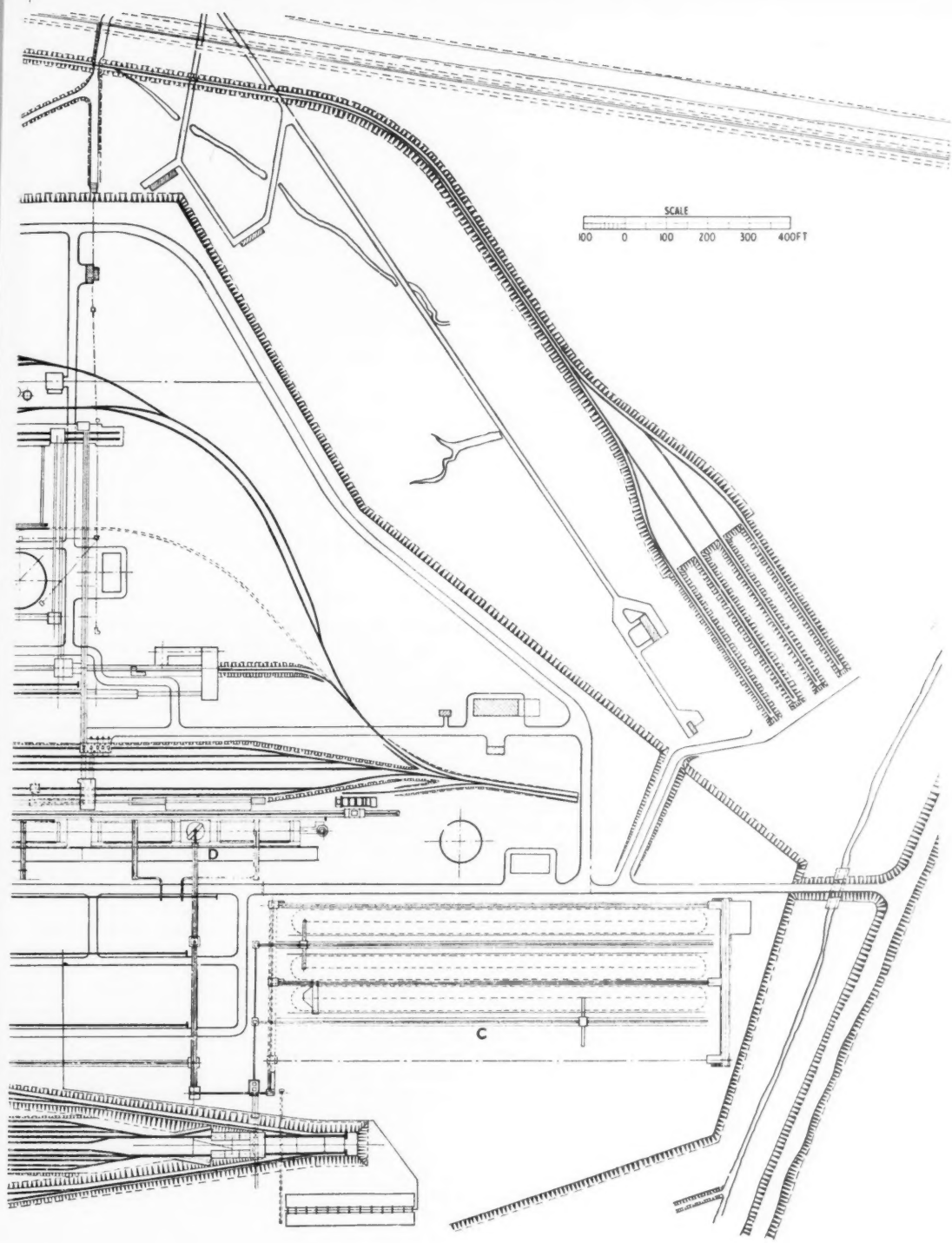
Manuscript received 7th September, 1953.

Mr. Jones and Mr. Walker are with John Summers and Sons, Ltd.



B Coal sidings E By-products plant F Blast-furnaces G Ore stockyard H Sinter plant J Power station

Fig. 1—Layout of blast-furnaces and coke ovens



C Coal stocking beds D Coke ovens

Fig. 1—Layout of blast-furnaces and coke ovens

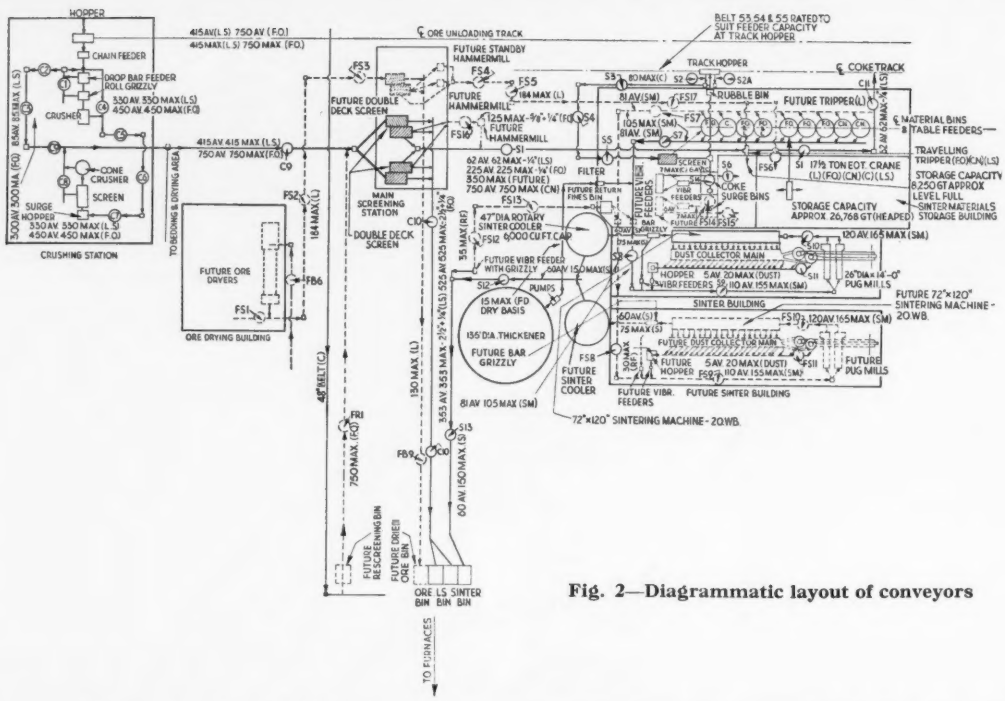


Fig. 2—Diagrammatic layout of conveyors

this hopper is suitably lined with manganese-steel wearing plates.

The drop-bar feeder is made of 7-in. dia. steel bars spaced at 9 in. centres mounted in manganese chains driven by cast-steel sprockets mounted on a head shaft, the centres of the driving sprockets being 11 ft. 8 in. and the working face of the feeder 5 ft. Material delivered to the feeder during its passage is scalped, the undersize material falling through the space between the bars. To prevent this material from sticking on the return strand as it is falling through the feeder, a drop-bar arrangement is included, whereby each alternate bar swings down below the adjacent bar in the line of the return belt. The feeder is fitted with a 2 to 1 speed range accommodated in the **V**-rope drive, and this enables the tonnage flow to be regulated to suit the delivery from the ore unloading hoppers.

From the drop-bar feeders the oversize material is controlled by a chain feeder so as to produce a steady flow of material on to the two-roll screen. The chain controller has a working width of 5 ft. and is driven by a 5-h.p. motor.

The material is further screened by a two-roll screen, which removes the $-2\frac{1}{2}$-in. material from the 500-tons/hr. feed to the primary crusher. The two-roll screen has a working width of 5 ft., a rear-roll diameter of 48 in., and a front-roll diameter of 30 in., the front roll being adjustable. The rear roll is grooved circumferentially, the grooves being 4 in. deep and about $2\frac{3}{4}$ in. wide at the periphery. The front roll is also grooved and these grooves are similar in size to those on the rear roll, but run longitudinally.

The primary crushing unit consists of belt-driven crushing rolls, 6 ft. dia. × 5 ft. wide. The roll shells are made of manganese steel and are toothed, one roll having ordinary teeth and the other slugger teeth. The manganese roll shells are mounted on cast-iron centres, which are in turn mounted on a forged-steel shaft, together with a wooden driving pulley with a toughened cast-steel centre. Each roll is driven by a 200-h.p. 585-r.p.m. motor through a flat-belt drive.

To assist in starting up the crushing rolls, a high-pressure oil-flotation system is fitted to the crusher bearings, and this floats the roll shafts until the rolls have begun to rotate. A special jacking device is incorporated in the crusher to assist in opening out the bearings. The crusher is capable of handling ore at the rate of 500 tons/hr. at 135 lb./cu. ft. with a roll setting of 6 in.

A four-roll feeder, with a working width of 3 ft., is installed in the discharge chute of the primary crusher, so as to break the impact of the product of the primary crusher on the collecting conveyor underneath, and also to feed this material on to the conveyor.

To collect material screened out by the drop-bar feeder and the two-roll screen, a 72-in. flat-belt conveyor, fitted with picking-type idlers, is installed in the feeder chute. This conveyor picks up the undersize material and delivers it either on to conveyor *C*4 (*see* Fig. 2), which later receives the primary crushed product, or by means of flapper gates and a by-pass chute on to the conveyor system which by-passes the secondary crusher.

A single roll feeder with a 3-ft. working width is installed in the chute to restrain the stream of material being fed on to conveyor *C*4 from the collecting conveyor *C*1. Similarly, the undersize material which has by-passed the secondary crusher is fed on to con-

veyor *C*8 by means of a single roll feeder. To reduce the load on the secondary crusher a 72 in. × 168 in. single-deck screen, fitted with a screen plate suitable for screening out material under 2½ in., is installed ahead of the secondary crusher. A standard 5½-ft. cone crusher is installed for secondary crushing, to reduce all material to −2½ in.

The gyratory crusher is fitted with coarse-crushing members and is mounted on a solid concrete foundation to absorb the loading. The crusher is driven by a 200-h.p. motor through a **V**-rope drive.

CRUSHER HOUSE

All the primary and secondary crushing equipment is installed in one crusher house, which is 123 ft. 6 in. long × 42 ft. span. The building layout is spacious and provides for the future installation of a further secondary crusher should this be required. Access to the building for crusher replacements is by rail track.

The crusher building is equipped with a 50-ton double-girder hand crane, the lifting capacity being given by two 25-ton crabs operating on the girders. A suitable lifting yoke is provided, so that when a lift in excess of 25 tons is required both hoisting units can be coupled together. Operation of the travel motions and the hoist is by hand from the crane operating platform, which is reached *via* the building steelwork.

SCREENING STATION AND STOCKYARDS

The crushed material is elevated to the screen house by conveyors and is then passed over a pair of 72 in. × 168 in. double-deck vibrating screens working in parallel. The top deck is arranged to screen out −1¼ in. material and the bottom to screen out −⅜-in. material. The oversize material from both decks is collected on conveyor *C*10 for delivery to the furnace highline. The undersize material from the screen is collected on conveyor *S*1*A* for delivery to the sintering plant. In the event of future installation of hammer mills an additional conveyor will be added and the oversize material from the bottom deck will be collected on the conveyor for tertiary crushing.

The building housing the final screens and chutes is 40 ft. × 36 ft. 6 in. × 72 ft. to the eaves. The layout of this building provides for the future installation of a tertiary crusher and also for the accommodation of a rescreening conveyor from the main ore stockyard.

Crushed and screened ore (and eventually sinter) are delivered into separate bins over one end of the highline ; these materials are then transferred to the furnace bins or the ore stockyard by two 50-ton capacity self-propelled transfer cars (*see* Fig. 3).

The stockhouse trestle situated to the north of the furnace is approached by a graded track. The trestle is over 1000 ft. long and comprises 30 2100-cu. ft. bins at 15 ft. centres, four 2400-cu. ft. bins at 17 ft. 6 in. centres, and two 7400-cu. ft. coke bins. The bins in 15-ft. bays are suspended in pairs, forming a double row, and the 17 ft. 6 in. bins are opposite the coke bins. The ore bins are of the parabolic type and are made of mild steel ; they are complete with double-lip type gates operated by the scale car. Coke and scrap bins are of the rigid type and the coke bins are lined with gunite.

The ore storage yard is about 1300 ft long by about 170 ft. clear width and has a capacity of about 200,000 tons of ore. The yard is serviced by a 34-ton ore bridge of 180 ft. span (*see* Fig. 3) ; this bridge is of the low girder type, the trolley running on top of the girders. The bridge is carried on a pier leg on the north side and on a shear leg on the south or furnace side. There are 16 travelling wheels. The rails for the shear leg are 30 ft. 5 in. higher in elevation than those for the pier leg.

Screened coke from the coke ovens is delivered direct by belt into the furnace coke bins. From the coke bins the coke is discharged on to two 60 in. × 120 in. single-deck vibrating screens, which are provided with a suitable mat for −1¼ in. screening. The oversize material from the coke screen is fed into two weigh hoppers, each of 250 cu. ft. capacity as-filled. These weigh hoppers are constructed of steel plate with manganese-steel wearing plates on the sloping bottom and have air-cylinder-operated gates for discharging the coke by chute directly into the blast-furnace skips.

Undersize material from the coke screen is elevated by either of two skip hoists working in parallel. The skip hoists, each with a capacity of 20 cu. ft., discharge the breeze into a bin positioned above the running track so that wagons may be loaded and this breeze removed.

For general cleaning duties in the skip pit a 7¼-cu. ft. capacity skip hoist is provided, which discharges into the appropriate furnace skip being loaded.

FURNACE CHARGING

For charging the furnace a double automatic skip is provided, as shown in Fig. 3 ; ore and limestone are charged into the skip cars from the scale car and coke is discharged automatically. The hoist is controlled by the scale-car operator ; once the skip has been set in motion the charging sequence is automatic.

The skip cars, each of 250 cu. ft. capacity, are of robust construction and run on cast-steel wheels mounted on forged-steel axles running on roller bearings. The skip-car body is lined with thick manganese liners.

The skip-hoist winch consists of a 6-ft. dia. double-grooved single drum, driven by two motors through double helical reduction gear. The skip bridge, which is of the through-truss type, pin-connected at the top and bottom, supports the double track for the skip cars. Details are as follows :

Bridge angle	59° ; 8 ft. 3 in.
Car travel	200 ft.
Max. rope speed	400 ft./min.
Weight of car	8 tons
Max. load	14 tons

The hoist house for the furnace is located underneath the skip bridge and is supported by steel framing over the hot-metal track. It accommodates the furnace-charging control equipment, skip-hoist winch and M G. set, pneumatic bell cylinders, and stockline recorders. The hoist house is air-conditioned.

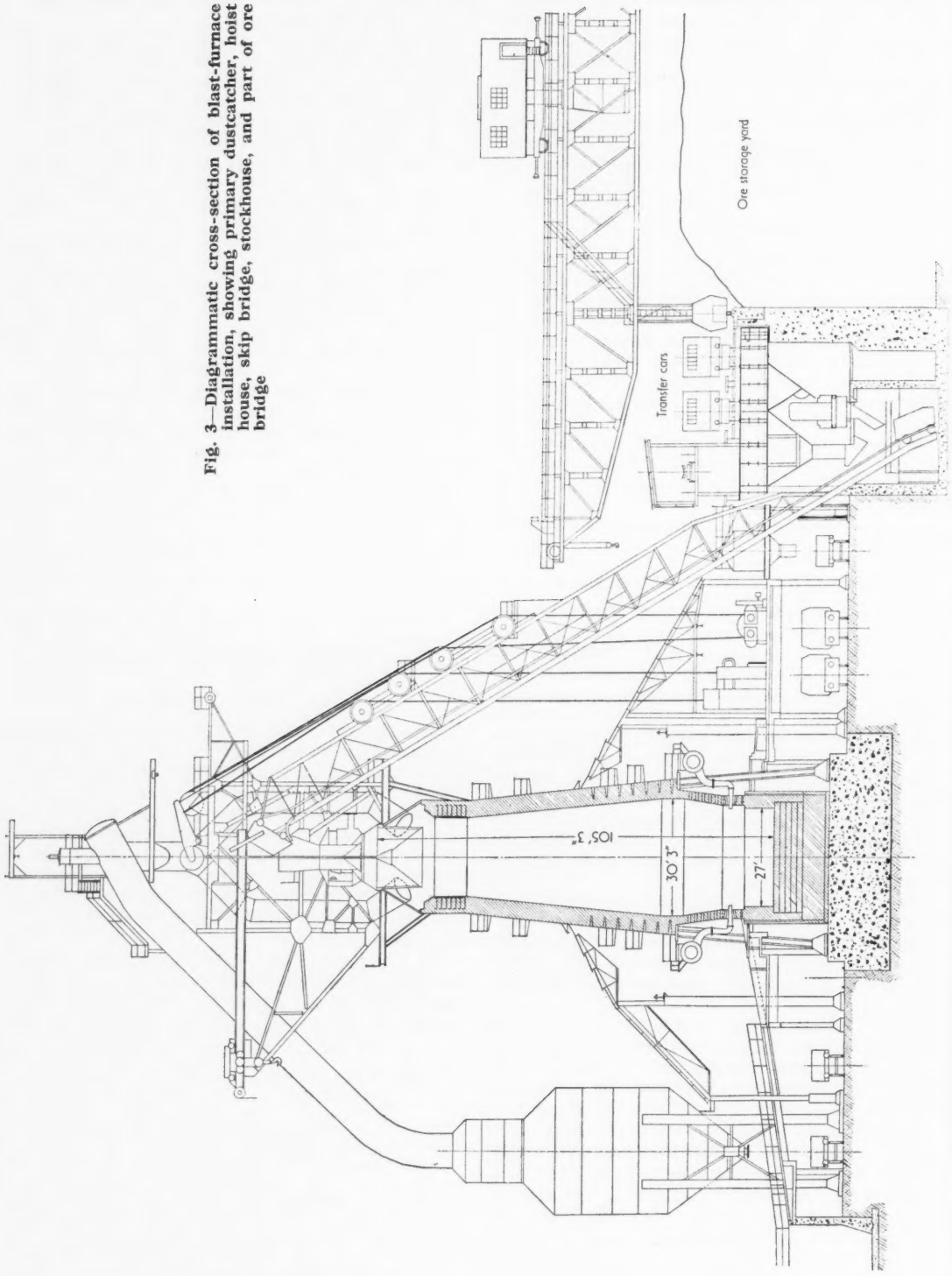


Fig. 3—Diagrammatic cross-section of blast-furnace installation, showing primary dustcatcher, hoist house, skip bridge, stockhouse, and part of ore bridge

The materials are charged into the furnace through an improved-type distributor (suitable for high-top-pressure operation). The revolving hopper (6 ft. 6 in. dia.) is supported on three rollers, held down by three rollers, and guided by three rollers, and it is provided with a manganese-steel bell seat. The small bell, which is in two halves, is 7 ft. dia. and is made of manganese steel. The distributor is driven by a D.C. motor through a worm reduction gear with selective reversing control and dynamic braking.

The small-bell hoist is pneumatic, being operated by the cold blast and, in the case of emergency, by the compressed-air system ; the operating cylinder is 33 in. dia. The large bell is a 15 ft. 6 in. dia. steel casting weighing 25 tons. The angle is 53° and the bell is completed with a cast-steel stiffening cone. The large hopper (15 ft. 10 in. top dia. with a 14 ft. 9 in. seat) is cast steel in one piece and weighs 33 tons. The large bell is operated by a pneumatic operating cylinder of 50 in. dia.

FURNACE HEARTH AND TOP

The furnace lines are shown in Fig. 4. The 27-ft. dia. hearth of the furnace is in firebrick, approx. 14 ft. thick, and is contained by a hearth jacket. The ring wall is made of carbon and the stack of firebrick. Suitable openings are provided in the carbon ring wall for tuyeres, coolers, iron notch, and two slag notches, the iron notch being 4 ft. below the slag notch.

Eighteen 6½-in. dia. copper tuyeres are spaced round the furnace between the nine supporting columns and are fed by the brick-lined bustle pipe (3 ft. 6in. inside dia. of brickwork).

The furnace stack is of riveted construction and sits on the mantle ring. The mantle ring in turn is supported by nine **H**-beam section columns which are of all-welded construction.

For cooling the hearth one set of hematite-iron segments is provided inside the hearth jacket. Eleven rows of copper coolers are installed for the bosh and seven rows of copper coolers for the stack, all being of the flat plate type. Water for hearth cooling is provided by an 8-in. circle pipe, and this is collected in a lower water trough positioned underneath the bustle main. A 14-in. circle pipe provides the water for bosh and stack cooling, and the overflow is collected in an upper water trough positioned above the bustle main.

The furnace-top structure, comprising a rigid structural steel frame independently supported from the top platform, gives access to a 55-ton trolley for removing the large bell and hopper during furnace relining and supports the receiving hopper platform, the bell-beam platform, and the bleeder-valve platform. There is a centralized lubrication system for the mechanical equipment on the furnace top, a separate automatic lubrication system being provided for the distributor seal.

CAST HOUSE

A cast-house building of heavy steel construction is provided for No. 1 furnace, provision being made for extension to accommodate No. 2 furnace at a later date. The cast-house floor supports the iron and slag runners from the furnace, these being sectional in hematite cast iron and completed with gates and operating levers. The clay-gun control house, constructed in brick with steel doors, is located on the south side of the cast house. Also accommodated in the cast house is a raw-clay bin for charging the electromechanical clay gun, together with miscellaneous material bins.

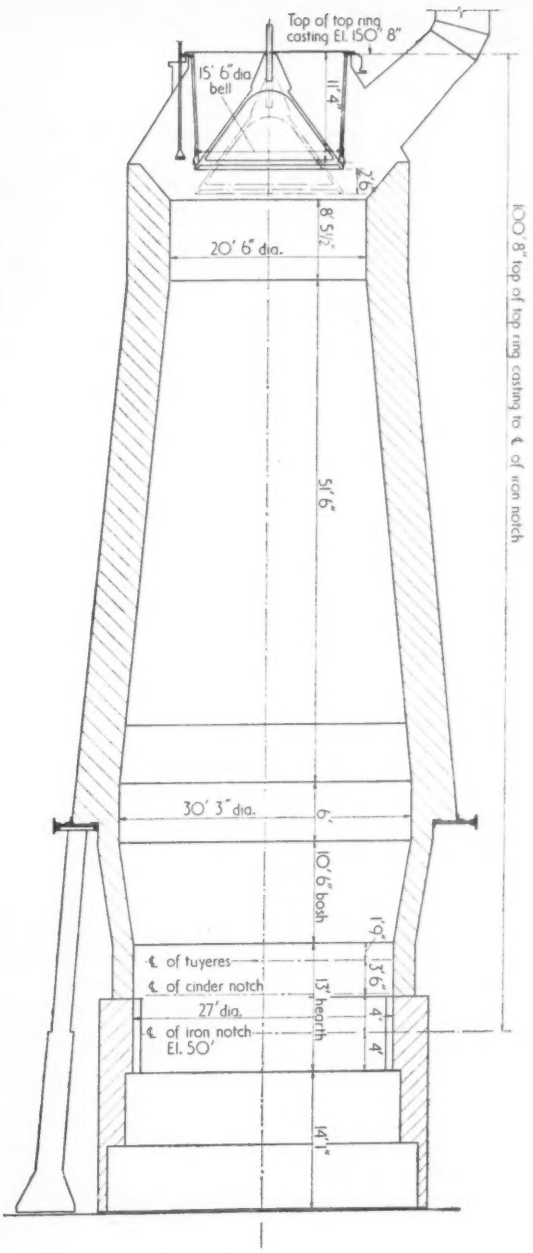


Fig. 4—Blast-furnace lines

Rail tracks to the cast house serve as slag and iron roads and also as a service track. The cast-house building is serviced by a 15-ton crane of 68 ft. span.

HOT-BLAST STOVES

Three hot-blast stoves are provided ; these are of the two-pass design with side combustion chamber, the heating surface provided in each being about 200,000 sq. ft. The shell is 26 ft. dia. × 102 ft. high, the checkers being made of nested tile. The combustion chamber is of elliptical construction, 55 sq. ft. in area. The ring wall of the stove is laid of stepped construction to suit the checker tile. Block insulation is provided between the stove shell and the ring wall and over the dome, and there is a layer of insulating brick underneath the combustion chamber.

One set of cast-iron columns with special alloy-iron caps and grids is provided for supporting the checker tile ; these grids and columns are capable of supporting the superimposed loads at temperatures of 800° F. There is a reinforced slab of insulating concrete in the bottom of each stove A pressure-type gas burner with a maximum capacity of 25,000 cu. ft. of gas per min., and complete with extendable sleeve, air draught with pulsation eliminator, motor-driven blower, and electric-driven regulator for combustion valve, is installed for heating The three stoves are connected by a common firebrick-lined flue, and the combustion gases are exhausted to atmosphere through a firebrick-lined brick stack, 10 ft. inside diameter at the top × 200 ft. high.

Cold blast from the turbo-blowers is led to the stoves by a 54-in. cold-blast main. The gas main at the stoves comprises a 48-in. header main reducing to 42 in at the stoves, with 30-in. dia. downlegs including water seal at each burner.

The hot-blast main from the stoves to the bustle pipe comprises a 72-in. main with three expansion joints. Stove operation is controlled from the stove platform, from which the burners and stove valves are accessible.

Each hot-blast stove has a stove-dome platform and the three are connected by walkways. These platforms are reached by an automatic electric passenger/goods lift, which also serves five other landings and has a car size of 5 ft. × 8 ft.

Raw gas taken from the furnace top is passed through uptakes and downcomer into the primary dustcatcher, which is of the expansion type, secondary cleaning being effected by a centrifugal cleaner. The primary dustcatcher is of all-riveted construction, lined with firebrick, with 31 ft. 0½ in. inside dia. of brickwork, and with carbon plate, the space between the carbon-plate liner and the shell being filled with insulating brick. The secondary dustcatcher shares a common support with the primary dustcatcher.

The exit from the secondary cleaner leads into a dust leg and from this the gas is passed into a washer, provision being made in the dust leg for the future connection from No. 2 furnace

The gas washer is a static tower of 18 ft. inside dia. ; it is of all-welded construction and is independently supported on four columns to permit easy replacement of shell plates in the event of corrosion. Inside the washer, three tile banks and one partition-ring bank are provided with suitable water sprays to ensure adequate wetted surfaces. There are wood liners around the inside of the shell to protect it against spray impingement. Effluent from the washer is discharged into an overhead flume for gravity flow to a thicker tank situated 800 ft. away.

Two wet-tube electrostatic precipitators are provided to fine-clean the washed gas. Each precipitator shell is of all-welded construction and accommodates 260 tubes 12 in. dia. × 15 ft. long, the inside diameter of the precipitator shell being 28 ft. The electrical equipment required for this precipitation is housed in a steel-framed brick building centrally located in the gas-cleaning plant. The ground floor of this building also accommodates all operating handwheels for the water-control valves in the gas-cleaning plant. The volume of gas cleaned in the cleaning plant from one furnace is 90,000 cu. ft./min. at 32° F. (dry). The dust content of the clean gas is as low as 0·005 grains/cu. ft.

Clean gas from the precipitator is stored in a 2,000,000-cu. ft. waterless gasholder, which is polygonal in shape, having 20 sides ; its diameter is 123 ft. 9 in. and the height to the eaves of the roof is 191 ft. 3 in. It is provided with an external and internal lift and there is a jib crane on the roof of the holder for maintenance purposes. The gasholder will maintain a static pressure of 12 in. W.G. in the gas mains. In the event of surplus clean gas being available, this can be bled off at a bleeder stack situated next to the precipitators, the amount of gas bled off being remotely controlled from the fuel control room.

SLURRY PLANT AND SLAG DISPOSAL

The effluent from the gas washer is delivered to a concrete thickener tank at a maximum rate of 6000 gal./min., and contains about 170 grains of dust per gallon (when two furnaces are working). The thickener mechanism, 135 ft. dia., is suitable for a single-compartment thickener with centre pier, and is complete with driving mechanism, lifting device, rakes, and valve assembly. The overflow water from the thickener goes to a sewer. The underflow water, containing about 45 tons of flue dust per day (dry basis) is pumped at 40–60 wt.-% of solids to a disc filter located in the sinter-materials storage building.

The vacuum disc filter is 8 ft. 6 in. dia. × 4 discs. For one furnace operation a detachable partition is provided between the second and third discs. The filter is capable of handling 75 gal. of thickener sludge per min., containing about 50 wt.-% of solids, and is complete with vacuum receiver, moisture trap, filtrate pump, vacuum pump, and rotary blower.

Slag is poured direct into a slag pit, from which it is taken by digger and dumper to the disposal point. The slag pit is divided longitudinally by a rough slag wall, and molten slag is poured into one half and watered and cooled while previously cooled slag is being removed from the other half.

SINTER PLANT

The sinter plant, which is not yet in operation, is designed for a maximum production of 1800 tons of

sinter per day on one machine ; provision is made for the future installation of a second strand.

The ore fines from the screening plant can be delivered into storage bins or into the storage yard. Coke breeze is delivered by means of a track hopper and conveyor system to a rod mill for crushing to $-\frac{1}{8}$ in. and is then stored in a special section of the storage yard. Flue dust is also handled by means of a track hopper and conveyors to storage bins. The storage yard provides storage for coke breeze and for seven types of ore and has a total capacity of 17,000 tons.

The sinter machine proper is a continuous type, 72 in. wide by about 120 ft. long with 20 wind boxes. The exhaust gases are drawn into a dust-collector main through downcomer pipes from the wind boxes. Four of these downcomers are fitted with Venturis. The exhaust gas from the main goes through two centrifugal dustcatchers, operating in parallel and each connected to one inlet of the double-inlet suction fan. Venturis are fitted in each of these connections. The fan is rated at 200,000 cu. ft./min., against 27 in. W.G. static pressure, and is driven by a 1550-h.p. 750-r.p.m. synchronous induction motor. Vane control gear is fitted in each fan inlet.

The hot sinter from the discharge end of the machine is passed under a disintegrator roll on to a hot screen of the vibrating type. The sinter fines through the screen go into a fines collecting bin, from which they are fed *via* a vibrating feeder to pug mills, which act as coolers.

The cooled fines are then taken by conveyor to a return-fines storage bin of 300 tons capacity, from which they are withdrawn as required by a table feeder and are then added to the other feed materials. The whole of the feed material is then passed through a drum-type primary mixer before being fed to the secondary pug-mill mixers at the charging end of the machine.

The sinter passing over the hot screen is discharged into a rotary-type cooler. This cooler has been modified for forced-air cooling, using cooling fans with an aggregate capacity of 200,000 cu. ft. of air per min. at 6–7 in. W.G.

FURNACE BLOWERS

Cold blast for the furnace is supplied by two turbo-blowers (one standby) located in the blowing and power station, which is about 400 yd. from the furnace. A third blower will be installed when the second furnace is built. The design duty of each blower is as follows :

Quantity of Free Air at 65° F. and 29·9 in. Hg, cu. ft./min.	Delivery Air Pressure, lb./ sq. in. gauge
66,700	20
80,000	30
68,500	35

The turbines are of the impulse-reaction type, comprising a two-row velocity followed by reaction blading. The maximum rating of the turbines is 8970 b.h.p. at 2740 r.p.m. They are of the straight condensing type, taking steam of 426 lb./sq. in. gauge and 730° F. at the stop valve, and are tapped for two-stage feed heating.

The blowers are five-stage single-flow, with radial-type curved-vane impellers. A non-return valve is fitted in the air-discharge main of each blower, and sluice valves in the air-blast mains are so arranged that any one blower may be coupled to any furnace. The air governing system comprises three regulators. One of these prevents surging of the machine, when only small quantities of air are required to be delivered, by opening a blow-off valve in a branch pipe from the discharge main and passing sufficient air to atmosphere to ensure that the delivery volume is maintained above the surging limit. A constant-volume regulator controls the air flow through the blower and maintains a constant air delivery, irrespective of variations in the burden resistance of the furnace. A constant-pressure regulator or limiting-pressure device is arranged to override the volume regulator should the furnace resistance build up to the limiting pressure for which the regulator is set. The last two regulators operate small servo cylinders incorporated in the turbine governor gear, and control is carried out by varying the speed of the turbine. The anti-surging regulator operates a butterfly-type blow-off valve through a servo cylinder and link gear, mounted on the valve body.

Each condenser for these turbines has a cooling surface of 5000 sq. ft., and is arranged three-flow. The auxiliaries include a two-stage air ejector and a two-stage extraction pump.

WATER COOLING

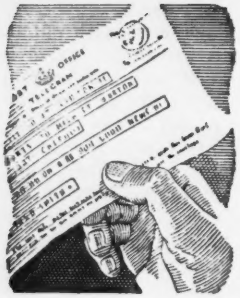
Cooling water for the furnace is in a closed system, being continuously circulated and cooled by means of tubular heat exchangers. The cooling medium in the heat exchangers is water from the Dee estuary. The circulating water is clarified and chemically treated so that there is very little possibility of deposition in the cooling members of the furnace. All the heat exchangers, pumps, etc., are located at the power station.

Water for the gas-cleaning plant is taken direct from the estuary, but a storage reservoir is incorporated in the system so that water of minimum salt content is used on the gas-cleaning plant. This is accomplished by filling up the reservoir when the salt content of the raw water is below certain limits, the stored water being sufficient to feed the plant during the remaining period of the tide cycle.

The furnace and the whole of the gas-cleaning plant have been designed for future high-top-pressure operation, and the whole system up to the final clean-gas main is capable of carrying up to 25 lb./sq. in. gauge pressure. The reason for this is the use of an expansion turbine to break down the pressure to the mains system figure of 12 in. W.G. which is imposed by the gas holder. By this means, not only will there be considerable recovery of power, but in addition the gas will be cooled and its water content will be reduced considerably.

Acknowledgment

The authors' thanks are given to the Chairman and Board of Directors of John Summers and Sons, Ltd., for their ready permission for this information to be published.



NEWS

ANNOUNCEMENTS AND NEWS OF SCIENCE AND INDUSTRY

THE IRON AND STEEL INSTITUTE

Autumn General Meeting, 1953

and

Symposium on Sinter

The **Autumn General Meeting** of The Iron and Steel Institute will be held at 4 Grosvenor Gardens, London, S.W.1, from **Tuesday, 24th November,** until **Thursday, 26th November, 1953.**

Associated with the meeting there will be a **Symposium on Sinter.**

Autumn General Meeting

Tuesday, 24th November

1.0–2.15 P.M.—Buffet Luncheon in the Library

Afternoon Session

2.15.–2.30 P.M.—Official business and election of Members

2.30–4.15 P.M.—Joint Discussion on:

"*The Effect of Emissivity and Flame Length on Heat Transfer in the Open-Hearth Furnace,*" by M. W. Thring (*Aug.*, 1952)

"*Experiments on Flame Radiation in an Empty Open-Hearth Furnace,*" by W. P. Cashmore and M. W. Thring (*Oct.*, 1953)

"*The Length of Oil and Gas Flames,*" by A. L. Cude (*Nov.*, 1953)

4.15–5.15 P.M.—Discussion on:

"*The Water-Cooling of Open-Hearth Furnaces,*" by A. M. Frankau (*May*, 1953)

Wednesday, 25th November

Morning Session

9.45–11.30 A.M.—Joint Discussion on:

"*The Notched Slow-Bend Test as a Brittle-Fracture Test,*" by J. E. de Graaf and J. H. van der Veen (*Jan.*, 1953)

"*The Fracture of Alpha Iron,*" by C. F. Tipper and E. O. Hall (*Sept.*, 1953)

"*Intergranular Brittleness in Iron–Oxygen Alloys,*" by W. P. Rees and B. E. Hopkins (*Dec.*, 1952)

"*The Cleavage Strength of Polycrystals,*" by N. J. Petch (*May*, 1953)

11.30 A.M.–12.0 NOON—Interval for light refreshments in the Library

12.0 NOON–1.15 P.M.—Discussion on:

"*Tensile and Impact Properties of High-Purity Iron–Carbon and Iron–Carbon–Manganese Alloys of Low Carbon Content,*" by N. P. Allen, W. P. Rees, B. E. Hopkins, and H. R. Tipler (*June*, 1953)

Symposium on Sinter

Wednesday, 25th November

1.0–2.15 P.M.—Buffet Luncheon in the Library

Afternoon Session

2.15–3.30 P.M.—Discussion on:

"*Sintering as a Physical Process,*" by H. Wendeborn (*Nov.*, 1953)

3.30–5.0 P.M.—Joint Discussion on:

"*The Mechanism of Reduction of Iron Oxides,*" by J. O. Edström (*Nov.*, 1953)

"*Softening of Iron Ores at High Temperatures,*" by A. Grieve (*Sept.*, 1953)

"*The Chemical Constitution of Sinters,*" by R. Wild (*June*, 1953)

"*Radiographic Studies of the Process of Sintering Iron Ores,*" by E. Cohen (*Oct.*, 1953)

Thursday, 26th November

Morning Session

9.45 A.M.–1.15 P.M. (Interval for light refreshments in the Library at about 11.0 A.M.)

—Joint Discussion on the following papers, introduced by Dr. T. P. Colclough, C.B.E. :

(Note: The papers will not be presented individually)

"*Factors Controlling the Rate of Sinter Production,*" by E. W. Voice, S. H. Brooks, W. Davies, and B. L. Robertson (*Oct.*, 1953)

"*The Permeability of Sinter Beds,*" by E. W. Voice, S. H. Brooks, and P. K. Gledhill (*June*, 1953)

"*Permeability Tests on Blast-Furnace Raw Materials,*" by E. W. Nixon and F. R. Maw (*Aug.*, 1953)

"*Effect of Mineral Additions and Moisture Control on the Sintering of Sierra Leone Concentrates,*" by P. K. Gledhill, G. C. Carter, and C. F. Ely (*Nov.*, 1953)

"*Sinter-Plant Assessment Trials at Dagenham and at Cleveland,*" by R. F. Jennings, E. W. Voice, P. K. Gledhill, G. C. Carter, and C. F. Ely (*Nov.*, 1953)

"*Towards Faster Sintering of Ironstone,*" by M. A. K. Grice and W. Davies (*Oct.*, 1953)

"*The Rating of Sinter Plants for Economic Output,*" by R. F. Jennings (*Nov.*, 1953)

1.15–2.15 P.M.—Buffet Luncheon in the Library

Afternoon Session

2.15–4.30 P.M.—Joint Discussion on:

"*Ironmaking from High-Sinter Burdens,*" by G. D. Elliot, J. A. Bond, and T. E. Mitchell (*Nov.*, 1953)

"*Sintering Practice at Domnarfvet, Sweden,*" by C. Danielsson (*Oct.*, 1953)

Presentations of Books

The Joint Library has recently received, through the courtesy of Mr. E. A. Woodward, Librarian at Messrs. Hadfields Ltd., Sheffield, two volumes and twelve pamphlets from the collection of the late Sir Robert Hadfield, published by the Russian Academy of Sciences on the occasion of the Mendeleev Jubilee Celebrations in Leningrad in 1924. The volumes comprise collected papers by Mendeleev on the periodic law and on investigations of aqueous solutions by specific gravity determinations, while the pamphlets consist of reviews of his work in various fields.

Dr. N. H. Polakowski has also kindly presented five recent Russian textbooks: "*The Metallurgy of the Light Metals*," by A. I. Belyaev (third edition, 1949); "*Modern Blast-Furnace Practice for the Specialist*," by N. I. Krasavtsev and M. Ya Ostroukhov (1949); "*Electric Melting Furnaces in Ferrous Metallurgy*," by N. V. Okorokov (third revised edition, 1950); "*The Preparation of Ores for Smelting*," by V. A. Kulibin (1952); and "*The Fundamentals of the Mechanization of Loading and Unloading Operations in Ferrous Metallurgy*," by K. A. Egorov (second revised and enlarged edition, 1952).

Volumes of *Teknisk Tidskrift* for the years 1893/4, 1895, 1906 to 1913, and 1915 to 1919 have been presented from the library of Imperial Chemical Industries Ltd., through the kindness of the Librarian, Miss Newton.

Acta Metallurgica

As previously announced, Members of The Iron and Steel Institute may subscribe to *Acta Metallurgica* at the privileged rate of £3 6*s*. 0*d*. per annum. It is emphasized that subscriptions will only be accepted if pre-paid, and that subscriptions at the privileged rate can only be accepted from individual Members of the Institute and not from firms or other organizations.

Applications must be made on the printed form of subscription, copies of which can be obtained from the Secretary, at 4 Grosvenor Gardens, London, S.W.1.

NEWS OF MEMBERS

▶ Mr. I. Codd has completed an Honours course in Metallurgy at the University of Leeds, and has joined the metallurgical staff of the English Steel Corporation Ltd.

▶ Mr. D. W. Edwards has left the United Steel Companies, Ltd., to join the Design Department of Garringtons Ltd., Bromsgrove, Worcs.

▶ Mr. D. J. I. Evans has been awarded the Ph.D. degree of the University of London, and has joined the staff of Sherritt Gordon Mines Ltd., Ottawa, Canada.

▶ Mr. A. F. C. Gardner has been appointed General Manager, North-Eastern Area, of Gibbons Bros. Ltd.

▶ Mr. L. C. Grimshaw has been appointed Manager of Steel Research to Firth Sterling Inc., McKeesport, Pa., U.S.A.

▶ Mr. W. A. Haven has retired from the Vice-Presidency of Arthur G. McKee and Co., Cleveland, Ohio, U.S.A.

▶ Mr. J. P. Hugo has been awarded the degree of Ph.D. (Met.) of the University of Sheffield.

▶ Mr. C. F. Hurst, Technical Director of Samuel Osborn and Co., has been appointed Deputy Chairman of the Company.

▶ Dr. T. Ko has left the U.K. to join the Institute of Metallurgical Research, Academia Sinica, Shenyang, China.

▶ Mr. D. S. McGregor has resigned his appointment as Head of the Department of Mechanical Engineering, Nottingham and District Technical College, to take up the post of Head of the School of Engineering, College of Technology, Leicester.

▶ Mr. J. J. Pick has been awarded the Ph.D. degree of the University of Birmingham and is now a Joseph Lucas Research Fellow in the Department of Industrial Metallurgy of the University.

▶ Mr. V. Pinney has emigrated to Canada, where he has joined the Metallurgical Department of the International Nickel Co., Ontario.

University of Sheffield

The United Steel Companies Research Fellowship in Metallurgy

Applications are invited for the above Fellowship tenable in the Department of Metallurgy of the University of Sheffield.

The object of the Fellowship is to advance knowledge in the field of Ferrous Metallurgy; candidates should already have some research experience, but not necessarily in this field.

Normal tenure will be five years, but the appointment will be for two years in the first instance and thereafter will be renewable annually.

Stipend £750 per annum for the first year, rising by annual increments of £50 to £950 per annum together with superannuation provision under the Federated Superannuation Scheme for Universities. A higher stipend may be paid to an exceptional candidate.

Applications (three copies) including names and addresses of referees and, if desired, copies of testimonials, should reach the Registrar, Dr. A. W. Chapman (from whom further particulars may be obtained), by not later than 18th November, 1953.

▶ Mr. S. V. Raman has left the U.K. and has joined Krauss-Maffei A.G., Munich-Allach, W. Germany.

▶ Mr. O. J. Thomas, F.I.M., a Director and Works Manager of the Glynhir Tinplate Co., Pontardulais, Glam., has been appointed General Manager.

▶ Mr. W. S. Turpin has left Grey and Marten Ltd. to join Hepworth and Grandage Ltd., Bradford.

Obituary

Mr. R. M. Bruce, of Wellington, New Zealand, on 10th June, 1953.

Wing-Commander E. Parrett, of Portsmouth, on 10th August, 1953.

Mr. Enos Smith, of British Ropes Ltd., Doncaster, on 18th August, 1953.

Mr. F. L. Tingle, Managing Director of the Tenuous Steel Co., Ltd., Sheffield, on 20th August, 1953.

BRITISH IRON AND STEEL RESEARCH ASSOCIATION

Staff

Mr. J. Savage, B.Sc., F.Inst.P., has been appointed Head of the Physics Department of the Association, in succession to Mr. M. W. Thring. Mr. Savage has been Head of the General Physics Section of the Department since 1947, and was appointed Deputy Head of the Department in 1948. He took up his new post on 1st October.

Instrumentation and Control of Mill Furnaces

A booklet entitled "*Instrumentation and Control of Mill Furnaces*" has recently been compiled by the Instruments Sub-Committee of the Plant Engineering Division of B.I.S.R.A. Its purpose is to stress the advantages of full instrumentation of rolling-mill furnaces, and to provide a greater understanding of automatic control systems, giving examples in each case.

The subject matter is divided into four sections: Measurement and controls for operation of mill furnaces; examples of soaking pit and reheating furnace instrumentation systems at present in operation; experiences and achievements in the use of automatic control; and installation and maintenance of works instruments. The booklet is obtainable from the British Iron and Steel Research Association, price 5*s*.

AFFILIATED LOCAL SOCIETIES

Ebbw Vale Metallurgical Society

The list of Officers for the Session 1953–54 is as follows :

President
A. W. Ellis, A.M.I.E.E.

Vice-President
J. J. Beese

Hon. Treasurer
T. Jones

Hon. Joint Secretaries
F. W. Lewis — W. B. Williams

Committee

W. H. R. Bird, M.A., B.Sc.	J. T. Mills
W. J. Davies	H. F. Padbury
J. K. Edwards, M.A.	G. R. Parkes, Ph.D.
P. Edwards	F. E. Probyn
I. Giddings	R. C. Powell, B.Sc.
D. A. Jenkins	J. F. Smith
W. D. Jenkins, B.Sc.	J. Thickins, B.Sc.
F. Llewellyn	W. E. Williams

Liverpool Metallurgical Society

The list of Officers for the Session 1953–54 is as follows:

President
H. J. Miller, M.Sc.

Vice-Presidents
G. T. Callis, B.Sc. — A. E. Griffin

Hon. Treasurer
R. S. Moore

Hon. Secretary
S. V. Radcliffe, B.Eng.

Committee

H. Edwards	W. Rose
T. H. Fletcher	B. Todd, B.Eng.
S. J. Kennett, B.Sc., Ph.D.	R. S. Brown, M.B.E.
E. C. Rollason, M.Sc., Ph.D.	V. L. Farthing

Newport and District Metallurgical Society

The list of Officers for the Session 1953–54 is as follows:

President
G. H. Latham, Ll.D., J.P.

Hon. Secretary
K. E. Thompson, B.Sc.

Hon. Treasurer
J. E. Wells, B.Sc.

Committee
T. Grey-Davies (*Chairman*)

H. R. Davies	R. A. Owen-Barnett, B.Sc.
E. D. Harry, B.Sc.	J. F. Pratt
B. Jones, D.Sc.	H. T. Roberts
C. I. F. Mackay	H. H. Stanley
C. J. Morgan	J. G. Vinsen, B.Sc.

Swansea and District Metallurgical Society

The list of Officers for the Session 1953–54 is as follows :

President
D. L. Blewitt, J.P.

Vice-Presidents
Prof. H. O'Neill, M.Met., D.Sc. — R. W. Evans, M.Met.

Hon. Treasurer
U. Williams

Joint Hon. Secretaries
H. A. Cooke — M. J. A. Thomas, B.Sc.

Council

R. G. Davies	A. J. K. Honeyman, B.Sc.
R. Walker	D. A. Bishop
J. S. Caswell, M.Sc.	R. J. Thomas
M. L. Hughes, Ph.D.	E. F. Meyer
W. R. Hitchings	D. J. Jones, B.Sc.

Manchester Metallurgical Society

The list of Officers for the Session 1953–54 is as follows:

President
W. B. Wragge, B.Sc.(Tech.)

Vice-Presidents
J. D. Hannah, M.Sc. — C. J. Bushrod, M.Sc.
G. A. Cottell, M.Sc.

Hon. Secretary
E. Holland, B.Sc.

Hon. Treasurer
N. Youatt

Council

K. M. Entwistle, M.Sc., Ph.D.	V. B. Hysel, M.Sc.
E. L. Francis, M.Sc.	J. Y. Davies, B.Sc.
F. H. Poole	W. Ashelby
R. H. Wallace	B. S. Berry, M.Sc.
C. F. Brereton	

Staffordshire Iron and Steel Institute

The list of Officers for the Session 1953–54 is as follows:

President
A. W. Shore

Vice-Presidents
K. G. Lewis, M.Sc., M.Sc.(Tech.) — A. Biddulph

Hon. Secretary
H. J. E. Jones

Hon. Treasurer
G. E. Lunt

Members of Council

W. Angus	D. Phimister
J. A. Davies	A. A. Shore
M. R. Liddiatt	N. F. Truman
L. V. Nicholls	W. M. S. Walker, Ph.D., M.Sc.
G. Parkin	W. H. Wesson
B. V. Peters	F. V. Wright

Members of Council (Past-Presidents)

W. Regan, B.Sc.	G. B. Morrison
K. H. Wright	R. G. Bashforth
W. M. Larke, B.A.	J. T. Wright

The Secretary of the Associates' Section is now Mr. N. F. Truman, 20 Church Road, Coseley, Staffs.

INSTITUTE OF METALS

Training of Metallurgists for Industry

With the authority of the Council of the Institute of Metals, the Metallurgical Engineering Committee has arranged to hold an Informal Discussion on "*The Training of Metallurgists for Industry*," at the Royal Institution, Albemarle Street, London, W.1, on Friday, 27th November, 1953. The morning session (from 10.30 a.m. to 12.45 p.m.) will be devoted mainly to the requirements of industry, and the afternoon session (from 2.15 to 4.30 p.m.) to attempts by the teaching profession to meet the needs of industry in this respect. The aim

of the Discussion is to consider the requirements and training of metallurgists for the whole of the metallurgical industry (extractive, ferrous, and non-ferrous); a few speakers representative of management and the teaching profession will be invited to make contributions, but otherwise it is intended that the Discussion shall be completely informal. Tickets will not be required for the meeting, but anyone wishing to attend should notify the Secretary of the Institute of Metals, 4 Grosvenor Gardens, London, S.W.1.

EDUCATION

Advanced Short Courses in Industrial Technology

Last year more than 500 advanced short courses, designed to enable scientists and technologists in industry to keep abreast of developments and new techniques, were provided by education authorities. The Ministry of Education believes that there is an urgent need to expand the provision of these courses, and has therefore asked Local Education Authorities, Regional Advisory Councils for Technical Education. and Regional Academic Boards to make a special survey, so that Local Education Authorities and Universities may make additional provision. The Ministry has also arranged to keep a list of all courses and to send particulars of them to the Federation of British Industries and the National Union of Manufacturers. Employing organizations and individuals are also invited to make suggestions for courses, either to the appropriate Regional Advisory Council or direct to the Ministry.

Courses for Public Schoolboys and University Undergraduates

At the invitation of the United Steel Companies, Ltd., 35 senior boys from 16 public schools spent a week in Sheffield as the guests of the Company and took part in a short course on iron and steelmaking. The aim of the course, which was held from 31st August to 4th September, was to give an insight into the manufacture of iron and steel and the manipulation of steel, the manufacturing uses of high-quality steels, and some characteristic problems in research.

A short course on iron and steelmaking, steelworks engineering, and research was also held from 7th to 12th September, for 34 undergraduates from 13 universities. The course, which was for students in Pure Science, Engineering, and Metallurgy, dealt with the scientific foundation of iron and steelmaking, certain derivative manufacturing processes, the scope of engineering in the industry, characteristic research problems, and the economic structure in the industry.

The object of both courses was to give the students an idea of the opportunities available in the iron and steel industry, with a view to aiding them in their choice of a career.

BRITISH IRON AND STEEL FEDERATION

Mr. Frederick Grant, Q.C., has been appointed Independent Chairman of the Executive Committee of the British Iron and Steel Federation, in succession to the late Sir Andrew Duncan.

NEWS OF SCIENCE AND INDUSTRY

Symposium on Cathodic Protection

The Corrosion Group of the Society of Chemical Industry will hold a Symposium on Cathodic Protection at the offices of the Institution of Electrical Engineers, Savoy Place, London, W.C.2, on Friday, 13th November, 1953. The Symposium will begin at 9.45 a.m. and will comprise seven papers on the protection of ships, underground pipes, storage tanks, etc., used in contact with water or soil in various parts of the world.

Conference on Welded Structures

A Conference on Welded Structures has been organized by a Joint Committee of the Ministry of Works, the Institution of Civil Engineers, and the Institution of Structural Engineers, and will be held on 23rd–26th November, 1953, at the offices of the Institution of Civil Engineers. The aim of the Conference is to review present knowledge and outstanding problems in the design, fabrication, and erection of welded steel structures. The discussion is arranged under eight headings: Multi-storey buildings; shipbuilding; dock gates and caissons; standing oil tanks and gas-holders; single-storey buildings; braced roof structures; bridges; and tubular construction. Further details and application forms may be obtained from the Joint Honorary Secretaries of the Conference, Institution of Civil Engineers, Great George Street, London, S.W.1.

Non-Ferrous Metal Industry in India

Under the auspices of the National Metallurgical Laboratory, Jamshedpur, it has been decided to hold a Symposium on "The Non-Ferrous Metal Industry in India" in January, 1954. The exact date and further details of the Symposium will be given in a later issue of the *Journal*.

Removal of Restrictions on the Use of Molybdenum

The Minister of Supply has issued a Direction removing restrictions on the use of molybdenum in alloy steel, as from 21st September, 1953. Restrictions on the use of nickel in alloy steel will continue in force for the time being.

Continuous Casting of Steel

A plant for the continuous casting of steel is to be installed in Sheffield by a group of steelmakers who are members of B.I.S.R.A. The plant will be constructed in the works of William Jessop and Sons Ltd., a member of the group, and the consulting engineers are Campbell, Gifford and Morton Ltd., London, S.W.1.

Scunthorpe Rod Mill Ltd.

The Board of Scunthorpe Rod Mill Ltd. has been rearranged following the transfer of the entire share capital of the Company to John Lysaght's (Scunthorpe Works) Ltd., and now consists of Sir Charles Bruce-Gardner (Chairman), N. R. R. Brooke, W. L. James, and K. S. Peacock.

International Conference on Hot Dip Galvanizing

The Zinc Development Association has published the official report of the Second International Conference on Hot Dip Galvanizing, which was held at Düsseldorf in June and July, 1952. The report, which contains the complete texts of papers presented at the Conference, is available direct from the Association or through booksellers, price 37*s*. 6*d*.

British Productivity Council

The British Productivity Council, which was formed in November, 1952, represents jointly management and

workers in all types of industrial activity. Its aims, which are set out fully in a new pamphlet entitled "*The British Productivity Council: Policy and Programme*", have as their object the improvement of productivity in every sector of the national economy by all possible means. To this end, the Council recommends the setting up of Local Productivity Committees in more than 100 towns and districts throughout the country. Committees would be made up of employers and trade union officials, as well as those concerned with technical education, and members of the local branches of professional institutions. Another scheme to be organized by the Council is the "Circuit Scheme for Exchange of Team Visits," in which firms are invited to enter teams. The pamphlet, which gives full details of these schemes, may be obtained from the British Productivity Council, 21 Tothill Street, London, S.W.1, price 6*d*.

Industrial Publications Received

▶ High Alumina Bricks and Cements—Technical data and applications—Pamphlet No. 2—John G. Stein and Co., Ltd.

▶ Notes on Machining the Nimonic Series of Alloys—Henry Wiggin and Co., Ltd.

▶ High Alumina and Fireclay Refractories—Publication No. 87 1½ M. M.P. 6/53: Heat and Acid Resisting Materials—No. 86 M.P. 1½ M/6/53—General Refractories, Ltd.

▶ Moisture and Mercury Vapour Traps—Brochure C.112/1: Vacuum Coating Plant—E.140/1: "Speedivac"; Rotary High Vacuum Pumps—A.120/1—W. Edwards and Co. (London), Ltd.

DIARY

3rd Nov.—Sheffield Society of Engineers and Metallurgists (Joint meeting with the Sheffield Metallurgical Association and the Institute of Metals)—"*The W1400 Walking Dragline*"—British Iron and Steel Research Association, Hoyle Street, Sheffield, 7.0 p.m.

10th Nov.—Liverpool Metallurgical Society (Joint meeting with the Institute of Welding)—"*The Development of Metallic Arc Welding and Electrodes during the past Half Century*," by E. Flintham—College of Technology, Byrom Street, Liverpool, 7.15 p.m.

11th Nov.—North Wales Metallurgical Society—"*Modern Lubrication Practice*," by H. J. Knight—County Primary School, Plymouth Street, Shotton, Chester, 7.15 p.m.

12th Nov.—Leeds Metallurgical Society—"*Zinc-base Die Casting Alloys*," by L. A. J. Lodder—Chemistry Department, The University, Leeds, 2, 7.15 p.m.

13th Nov.—Society of Chemical Industry (Corrosion Group)—Symposium on Cathodic Protection—Institution of Electrical Engineers, Savoy Place, London, W.C.2, 9.45 a.m.

13th Nov.—Australian Institute of Metals (Melbourne Branch)—Symposium on the Theory and Practice of Wire Drawing—Melbourne, Australia.

16th Nov.—Institution of Engineering Inspection (Wolverhampton Branch)—"*Precision (Lost Wax) Castings*," by W. N. Jones—Compton Grange, Compton Road, Wolverhampton, 7.30 p.m.

16th Nov.—Sheffield Society of Engineers and Metallurgists (Joint meeting with the Sheffield Metallurgical Association and the Institute of Metals)—"*Indentation Hardness*," by D. Tabor—The University Building, St. George's Square, Sheffield, 7.30 p.m.

19th Nov.—Liverpool Metallurgical Society—"*Metallurgical Problems connected with the Development of Atomic Energy*," by L. Rotherham—Liverpool Engineering Society, The Temple, Dale Street, Liverpool, 7.0 p.m.

19th Nov.—North East Metallurgical Society—Discussion on "*The Training of a Metallurgist*"—Cleveland Scientific and Technical Institution, Middlesbrough, 7.15 p.m.

20th Nov.—West of Scotland Iron and Steel Institute—"*The Development of Basic Open-Hearth Practice during the last Forty-five Years*," by J. A. Anderson—39 Elmbank Crescent, Glasgow, 6.45 p.m.

20th–21st Nov.—Institute of Physics (X-ray Analysis Group)—Autumn Conference—London.

23rd–26th Nov.—Welded Structures Conference. Organized by a Joint Committee of the Ministry of Works, the Institution of Civil Engineers, and the Institution of Structural Engineers—Institution of Civil Engineers, Great George Street, London, S.W.1.

24th–26th Nov.—The Iron and Steel Institute—Autumn General Meeting and Symposium on Sinter—4 Grosvenor Gardens, London, S.W.1.

25th Nov.—Royal Institute of Chemistry (London Section)—"*Corrosion in Action*," by E. J. Vaughan—Technical College, Lowfield Street, Dartford, 6.30 p.m.

25th Nov.—Institution of Engineering Inspection (West of Scotland Branch)—"*Non-Destructive Testing of Alloy Steels*," by J. Cameron—St. Enoch Hotel, Glasgow, 7.30 p.m.

25th–26th Nov.—Institution of Gas Engineers—18th Autumn Research Meeting—London.

26th–29th Nov.—Ceylon Association for the Advancement of Science—9th Annual Session—Colombo.

27th Nov.—Institute of Metals—Discussion on "*The Training of Metallurgists for Industry*"—Royal Institution, Albemarle Street, London, W.1, 10.30 a.m. and 2.15 p.m.

TRANSLATION SERVICES

(The previous announcement was made in the October, 1953, issue of the *Journal*, p. 216).

Translations Available

No. 476 (German). E. Schauff: "Critical Comparison of the Heat-Treatment Plants in Cold-Rolling Mills." (*Stahl und Eisen*, 1953, vol. 73, July 2, pp. 895–902).

No. 477 (French). A. Beneteau: "Theory and Practice of Automatic Looping in Rolling Mills." (*Révue de Métallurgie, Mémoires*, 1953, vol. 50, Apr., pp. 229–247).

Translation in Course of Preparation

(Russian). O. A. Esin and A. I. Okunev: "The Kinetics of the Interaction of Metal and Slag from the Point of View of the Ionic Theory." (*Izvestiya akademii Nauk SSSR, Otdelenie Tekhnicheskikh Nauk*, 1952, No. 10, pp. 1472–1482).

Charges for Copies of Translations.—The charge for translations is £1 for the first copy and 10*s*. for each additional copy of the same translation. Requests should be accompanied by a remittance. These translations are not available on loan from the Joint Library.

Translations Prepared at Members' Requests—Members requiring translations of foreign papers are invited to communicate with the Secretary, who will ascertain whether they can be prepared for inclusion in the Series.

ABSTRACTS AND BOOK NOTICES

IRON AND STEEL MANUFACTURE AND RELATED SUBJECTS

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MINERAL RESOURCES

On the Genesis of Magnesite and Talc Deposits in the Eastern Alps. O. M. Friedrich. (*Radex Rundschau*, 1951, No. 7, Nov., 281–298). A survey of literature is presented on deposits of talc and magnesium carbonate in the eastern Alps. Mode of formation and relationship to iron carbonate deposits are discussed. (77 references).—E. C.

Do Iron Ore Deposits Exist in the Continental Submarine Shelf of Biscay ? F. Costa. (*Bol. Min. e Indust.*, 1952, **31**, Dec., pp. 507–509). [In Spanish]. The author discusses the possibility of the existence of ore in the submarine platform off the Bay of Biscay. No data are definitely known yet but the investigations of the possibilities is an urgent problem.

The Iron Ore Deposits of the United States and Canada. J. Bertin-Roulleau. (*Mét. Constr. Mécan.*, 1953, **85**, Apr., 307–309). A short review of the known iron ore deposits in the U.S.A. and Canada is made.—B. G. B.

Mineral Supplies and Their Measurement (Iron-Manganese) A. Legendre. (*Proc. United Nations Sci. Conf. on the Conservation and Utilization of Resources*, Lake Success, 17 Aug.–6 Sept. 1949, **2**, *Mineral Resources*, 4–11). The reserves of manganese, which are concentrated in the hands of a few important producers, are not in danger of rapid exhaustion. The iron ore position is much more serious, and the examples of American and Lorraine ores are cited. Solutions for the problem of utilizing the siliceous ore which is likely to outlast the calcareous ore have not been fully worked out.—J. C. B.

ORES—MINING AND TREATMENT

Mining in the Basque Country. G. Vié. (*Echo Mines*, 1953, Mar., 183–186). An account is given of the development of certain iron mines in the Aldudes mountains and of research into the nature of the deposits.—A. G.

Bethlehem Steel's New Grace Mine. A. W. Knoerr. (*Eng. Min. J.*, 1953, **154**, Jan., 88–90, 109). Details are given of shaft sinking and progress of other installations. The blind orebody (depth 1500–3000 ft.) was located by airborne magnetometric survey. Water in the shafts is combatted by pumping to lower the water-table. Output is planned at 12,000 tons/day of ore and waste : the ore, of average 44% iron, will be concentrated to 65%.—K. E. J.

Noise in Mines. B. Johansson. (*Jernkontorets Ann.*, 1952, **136**, 11, 468–508). [In Swedish]. The results are reported of investigations, in the Grängesberg iron ore mine and in an echo-free room, of noise arising from different mining machinery, its effect on miners' hearing, methods of reducing the sound level at the source, and ear protection. Hearing loss measurements were made on 203 workers, the effects of three weeks' holiday on six miners' hearing were determined, and the different acoustic losses for nine types of ear plug were measured with an audiometer. —R. A. R.

Radioactive Isotopes in Mineral Dressing Research. J. S. Carr. (*Atomic Energy Research Establishment Report* CE/R 912 : Ministry of Supply, 1952). The principles of radioactive techniques are reviewed first. A detailed examination of the use of radioactive isotopes in the study of the mechanism of the flotation of minerals is made. (99 references).—B. G. B.

New Results on the Pelletizing of Iron-Ore Concentrates. G. Sengefelder. (*Iron Steel Inst. Translation Series*, 1953, Sept., No. 473). This is an English translation of a paper which appeared in Stahl u. Eisen, 1952, **72**, Dec. 4, 1577–1579 (*see* J. Iron Steel Inst., 1953, **174**, June, 164).

Sintering and Melting Points of Iron Ores from New Quebec. J. Laneuville. (*Province of Quebec, Canada, Dept. of Mines, Laboratories Branch*, 1952, P.R. No. 265).

Disintegratability of Iron Ores from New Quebec Compared with Standard Ores. J. Laneuville. (*Province of Quebec, Canada, Dept. of Mines, Laboratories Branch*, 1952, P.R. No. 276).

Porosity and Adsorptivity in Iron Ores from New Quebec Compared with Standard Ores. J. Laneuville. (*Province of Quebec, Canada, Dept. of Mines, Laboratories Branch*, 1953, P.R. No. 277).

Ironmaking from High-Sinter Burdens. G. D. Elliot, J. A. Bond, and T. E. Mitchell. (*J. Iron Steel Inst.*, 1953, **175**, Nov., 241–247). [This issue].

Sinter-Plant Assessment Trials at Dagenham and at Cleveland. R. F. Jennings, E. W. Voice, P. K. Gledhill, G. C. Carter, and C. F. Ely. (*J. Iron Steel Inst.*, 1953, **175**, Nov., 267–277). [This issue].

Effect of Mineral Additions and Moisture Control on the Sintering of Sierra Leone Concentrates. P. K. Gledhill, G. C. Carter, and C. F. Ely. (*J. Iron Steel Inst.*, 1953, **175**, Nov., 277–279). [This issue].

The Rating of Sinter Plants for Economic Output. R. F. Jennings. (*J. Iron Steel Inst.*, 1953, **175**, Nov., 248–256. [This issue].

Sintering as a Physical Process. H. B. Wendeborn. (*J. Iron Steel Inst.*, 1953, **175**, Nov., 280–288). [This issue].

The Mechanism of Reduction of Iron Oxides. J. O. Edström. (*J. Iron Steel Inst.*, 1953, **175**, Nov., 289–304). [This issue].

Sinter Sampling Technique Developed by U.S. Steel Corp. R. L. Stephenson and D. J. Carney. (*J. Met.*, 1953, **5**, May, Section 1, 628–629). The method of sampling and testing sinter at the United States Steel Corp. is briefly described. Sampling is done by a steel framework basket which travels the full length of the sintering machine.—G. F.

The Magnetizing Roasting of Iron Ores. H. Kirchberg. (*Met. u. Giesserei Techn.*, 1952, **2**, Dec., 424–427). This is a survey of methods of magnetizing roasting, with particular reference to the principles, methods of operation, and results obtained with the Lurgi process and the process developed by the Kaiser-Wilhelm-Institut für Eisenforschung.—L. J. L.

Beneficiation of East Texas Iron Ores Basis of Lone Star Steel Operations. W. R. Bond. (*J. Met.*, 1953, **5**, Apr., 515–519). The iron ore holdings of the Lone Star Steel Co. in Texas are described, and the development of beneficiation processes which have resulted in successful blast-furnace operation is discussed. The main features are dehydration of the limonite, calcination of the siderite, washing processes enabling large tonnages of low-grade material to be handled, and reclamation of fines.—G. F.

A Method of Evaluating the Mechanical Properties of Heavy Suspensions Used for the Separation of Minerals According to Their Specific Gravity. I. Z. Margolin. (*Doklady Akademii Nauk S.S.S.R.*, 1953, **89**, 1, 117–120). [In Russian]. The method in which the resistance encountered by fine grains of ore slowly moving in a suspension is simulated by a thin plate whose specific surface is equal to that of the grain under consideration is described. By comparing the effective specific gravity of the plate with that of the heaviest piece of rock floated in a separator, it is possible to evaluate the degree of agreement between calculated and observed resistance to displacement and to predict the results of ore separation in a given suspension.—V. G.

A Little Chemistry for the Flotation Plant Worker. K. Hillmeyer. (*Met. u. Giesserei Techn.*, 1952, **2**, Oct., 353–354). Explanations are given of terms relating to chemical reactions in flotation processes.—R. A. R.

Upgrading Domestic Manganese Ores by Leaching with Caustic Soda. R. V. Lundquist. (*Min. Eng.*, 1953, **5**, Apr., 413–417). Investigations on a range of manganese ores and concentrates show that many can be considerably beneficiated by treatment with NaOH to remove SiO_2 : opaline varieties of SiO_2 respond most readily. Spent liquor can be reactivated by a lime treatment.—K. E. J.

FUEL—PREPARATION, PROPERTIES, AND USES

Heat Flow Diagrams as a Means of Setting up Technically Justified Fuel Consumption Standards. H. Sommer. (*Technik*, 1953, **8**, Jan., 17–22). Having explained the principle of the Sankey diagram, the writer illustrates its use in the presentation of heat balances for a rotary sinter furnace, blast-furnaces, an open-hearth furnace, and a modern boiler plant.—J. G. W.

The Solution of Transient Heat-Flow Problems by Analogous Electrical Networks. D. I. Lawson and J. H. McGuire. (*Proc. Inst. Mech. Eng.* Preprint, 1953, July 18). The design of a machine for solving heat-conduction problems by electrical analogy is described. The apparatus has been designed to give cathode-ray-tube presentation of the results for problems in which the boundary condition at the heated surface is a prescribed time-temperature curve. A novel feature of the apparatus is the introduction of non-linear circuit elements, to simulate the loss of heat by radiation and convection cooling.—P. M. C.

The Length of Oil and Gas Flames. A. L. Cude. (*J. Iron Steel Inst.*, 1953, **175**, Nov., 304–312).

Some Raw Material Problems of the Indian Iron and Steel Industry. P. Kutar. (*Indian Inst. Met., Presidential Address, 6th Annual General Meeting*, 1952 ?). Work carried out on coal washing is described. As the iron ores used are very rich, the coal-washing treatment is intended to obtain a product with a consistent ash content of about 15%. The high Al_2O_3/SiO_2 ratio of the gangue in many of the iron ores leads to difficult blast-furnace operation so that beneficiation experiments are being carried out to reduce the Al_2O_3 content. The high cost of oxygen is restricting its use in the Indian steel industry.—B. G. B.

A New Self-Sealing Coke-Oven Door. G. Cellan-Jones. (*Coke Gas*, 1953, **15**, June, 221–222). A description is given of a German Patent whereby the firebrick door plugs are provided with a system of vertical and horizontal channels to remove the gas formed at the oven ends to the oven crown. This prevents the formation of tarry deposits in the door fittings and removes the necessity for continual cleaning to ensure proper functioning of the doors.—B. C. W.

Coals for Carbonization and Gasification in Scotland. E. A. C. Chamberlain. (*Iron Coal Trades Rev.*, 1953, **166**, Apr. 3, 761–765). The author discusses the problem of the supply of coking coals as it concerns the Scottish industry, and outlines the factors which must be considered in planning the future of the various types of coal.—G. F.

Coke Breeze and Coke in Western Germany. M. P. Boutet. (*Rev. Ind. Min.*, 1953, **34**, May, 395–404). Large amounts of Lorraine coal are now utilized in the manufacture of metallurgical coke by the addition of coke breeze imported from Germany (1 ton breeze to 3 tons coal). A study of future requirements of both France and Germany indicates that Germany will not be able to supply sufficient coke breeze for this purpose, and the importance of reducing the coke addition to a minimum is stressed.—A. G.

The Petrographic Analysis of Coals and Its Relation to Coke Making. B. Alpern. (*Rev. Ind. Min.*, 1953, **34**, May, 359–373). After a description of the classification and petrographic nomenclature of coal constituents, methods of preparation and examination of coals and cokes are given. Inert materials are eliminated by dense-media separation and a quantitative study of the sample set in resin is made using an integrating stage.—A. G.

New Ideas on Coal Preparation. (*Coke Gas*, 1953, **15**, May, 175–178). The principles of coal preparation and blending before carbonization are outlined. Particular attention is given to the Burstlein process in which the harder fraction of the coal blend is ground to small size, resulting in a more uniform coke of improved quality.—T. McK.

Coke from Weakly-Coking Coals. L. T. Minchin. (*Coke Gas*, 1953, **15**, May, 167–171). Methods employed in Lorraine to enable the local high-volatile coals to be used to make metallurgical coke are described. Processes evolved include blending with coal of high bitumen content and coke breeze ; using low-temperature char as part of the blend ; and crushing the durain constituent to small size.—J. McK.

A New Coal Blending Plant. (*Steam Eng.*, 1953, **23** June, 326–328). **Shotton Coking Plant.** (*Coke Gas*, 1953, **15**, Apr., 117–125 ; May, 155–161). **Coal Blending: New Plant Installed at Shotton.** (*Iron Coal Trades Rev.*, 1953, **166**, May 15, 1117–1119). The layout and operation of the new coal-blending plant for the coal-carbonizing section of John Summers and Sons Ltd. are described. The plant has a capacity of 300 tons/hr. and is claimed to be the first in the world to use the Robins-Messiter system of bedding and blending.—G. F.

The Work of the Association on Coking Pressure. W. Idris Jones. (*Brit. Coke Res. Assoc. 5th Conf.*, June 26, 1952, 2–16). The movable-wall oven as operated under standardized conditions is capable of yielding repetitive and interpretable results concerning coking pressure. The effect of variation in bulk density, rate of heating, and blending on coking pressure, is described. Bulk density of the charge is of great importance in determining coking pressure. The results of work on the measurement of internal pressure are discussed.

The Application of the Results from the Point of View of Oven Construction and Life. T. C. Finlayson. (*Brit. Coke Res. Assoc. 5th Conf.*, June 26, 1952, 17–19). The application of the results of work carried out on the movable-wall oven to the operation and design of full size ovens is discussed.

The Mechanism of Coke Formation and the Development of Coking Pressure. H. E. Blayden. (*Brit. Coke Res. Assoc. 5th Conf.*, June 26, 1952, 20–23). The author shows how the established theory of coking can explain, at least qualitatively, the development of coking pressure.—B. G. B.

On the Production and Uses of High-Temperature Lignite Coke. G. Bilkenroth and E. Rammler. (*Met. u. Giesserei Techn.*, 1953, **3**, Feb., 61–67). The principles and practice of the high-temperature coking of brown coal are discussed.

Developments in the Recovery of By-Products from Coke-Oven Gases. C. L. Boucher and D. Bagley. (*Society of Engineers*, 1953, **44**, Jan.–Mar., 15–33). A description of modern plant for the recovery of by-products from coke-oven gas is given.—B. G. B.

The Metallurgical Coke Outlook. (*Coke Gas*, 1953, **15**, May, 151–152). A brief note is given on the coke requirements of the iron and steel industry.—J. McK.

Automatic Control for Gas Producers. (*Gas World*, 1953, **137,** June 20, 1727). Details are given of a control scheme developed by George Kent Ltd. and the British Coal Utilisation Res. Assoc. for the automatic control of steam, air, and coal feed, in a gas producer associated with open-hearth furnaces.—B. C. W.

The Operation and Control of Gas Producers for Use in Steelworks. P. Devilder. (*Mét. Constr. Mécan.*, 1953, **85,** Jan., 47–53). The operation of gas producers to give a gaseous fuel for open-hearth furnaces is considered. The use of instruments (*eg.*, CO_2 recorders) to control and improve the operation of gas producers is discussed.—B. G. B.

TEMPERATURE MEASUREMENT AND CONTROL

Control of Heat Treating Furnaces Improves Quality. L. Walter. (*Steel Processing*, 1953, **39,** Apr., 177–181, 190). The principles of simple on-off control, and the more effective multi-step or "high-medium-low" system are discussed. Details of suitable temperature control methods for electric, gas, and oil-fired furnaces are given.—P. M. C.

A Photo-Electric Temperature Controller. K. Allgeuer. (*Radex Rundschau*, 1952, No. 2, Apr., 93–94). A concise description is given of a photo-electric temperature controller for furnaces which is claimed to be capable of close accuracy at both high and low temperatures.—E. C.

Theoretical and Practical Determination of Combustion Temperatures. G. Hofmann. (*Met. u. Giesserei Techn.*, 1952, **2,** Dec., 428–432). A survey of the theoretical methods of calculating combustion temperatures is followed by notes on their practical determination. Several formulæ are given and explained.—L. J. L.

The Temperature of Combustion of Industrial Fuels in Air. E. Schwarz-Bergkampf. (*Radex Rundschau*, 1951, No. 7, Nov., 265–280). A number of graphs and nomogrammes are computed for the calculation of combustion temperatures for high-grade and low-grade solid and gaseous fuels. The graphs are explained and several examples are given to demonstrate practical applications.—E. C.

Visible Temperature Indication in Industrial Furnaces. F. Drexler. (*Chemiker Zeitung*, 1953, **77,** Apr. 5, 205–208). Practical examples are given of the use of temperature recording paints called 'Thermocolor' containing an organic pigment that changes colour on being heated to a pre-determined temperature. These paints can be painted or sprayed on the furnace and will give rapid visual indication of the temperature distribution. It is claimed that the temperature indications are accurate within $\pm 5°$ C.—L. J. L.

Gas Temperature Measurement in Dusts. G. S. Cantle and E. P. Hotchen. (*Steam Eng.*, 1953, **22,** Apr., 247–249). Temperature traverses were taken at two sections of a horizontal duct carrying hot gas, the flow of which could be controlled by dampers. The influence of a bend in the duct, of the use of a rotary damper, and of variations in rate of gas flow on the stratification of the gas was investigated.—J. MCK.

REFRACTORY MATERIALS

Sintered Dolomite for Steel Making. S. Pawlowski. (*Hutnik* (Katowice), 1953, **20,** 3, 103–109). [In Polish]. Processes taking place during the firing and sintering of dolomite and the properties required from the sintered material are outlined. Data on the Polish production of sintered dolomite are given.—V. G.

Properties and Uses of Semisilica Brick. G. B. Remmey. (*Iron Steel Eng.*, 1953, **30,** May, 94–100). The behaviour of semisilica brick in the A.S.T.M. Panel Spalling test is described and the structural spalling and protective glaze theories are discussed. This brick, with 76 to 78% SiO_2, it is claimed, has high resistance to structural spalling, good resistance to thermal spalling, and a low resistance to plastic flow.

The Testing and Development of Basic Refractories for the Open Hearth. L. W. Austin. (*Blast Furn. Steel Plant*, 1953, **41,** June, 639–645). Details are given of a gas-fired furnace for the testing of refractory bricks at high temperatures (above 1700° C.). The test procedure adopted is described.

Relationships between Particle Size and Energy Content of Magnesium Oxide. B. Weissenbach. (*Radex Rundschau*, 1951, No. 6, Oct., 257–260).

The Adhesion of Magnesite Powders to Solid Surfaces. I. F. Conrad, E. Cremer, and T. Kraus. (*Radex Rundschau*, 1951, No. 6, Oct. 227–233). The concepts and principles of adhesion between solids are explained and a method is described for examining the relationships between mass of powder, adhesive force, adhesive tension, and friction. It is shown that only fine-grained powders, up to a definite size limit, are affected by adhesive forces, friction being the over-riding factor with coarser powders.—E. C.

Electrical Conductivity of Silicates in Relation to Temperature and Composition. K. H. Schwarz. (*Radex Rundschau*, 1951, No. 7, Nov., 299–306). The investigation of electrical conductivity of a series of lithium-aluminium silicates is described. Specific resistance is found to remain high during heating but drops to a few ohms at the melting point. Lithium-rich silicates are better conductors than aluminium-rich ores, ortho-silicates are better than metasilicates, and eutectics better than solid solutions.—E. C.

A Contribution to the System Al_2O_3-Fe_2O_3-SiO_2. H. Nowotny and R. Funk. (*Radex Rundschau*, 1951, No. 8, Dec., 334–340). The preparation of binary and ternary compounds of this system is described and results obtained by X-ray powder photograph are discussed. The results of magnetic measurements are used to supplement the X-ray data in view of the dissociation of Fe_2O_3 at higher temperatures. Curie Points were also determined.—E. C.

The Fusion Diagram of the CaO-CaF_2 System. P. P. Budnikov and S. G. Tresvyatskii. (*Doklady Akademii Nauk S.S.S.R.*, 1953, **89,** 3, 479–482). [In Russian]. Melting temperatures of lime-fluoride mixtures were determined using seger cones and thermal analysis.—V. G.

The Fusion of Mullite. N. E. Fiolenko and I.V. Lavrov. (*Doklady Akademii Nauk S.S.S.R.*, 1953, **89,** 1, 141–142). [In Russian]. The decomposition of mullite on melting at 1830° C. with the formation of corundum and a silica-rich liquid was experimentally demonstrated.—V. G.

Spalling-Resistant Magnesite Refractories. F. Nadachowski. (*Prace Instytutow Ministerstwa Hutnictwa*, 1953, **5,** 2, 105–112). [In Polish]. Factors influencing the resistance of refractory materials to thermal shock are discussed. Investigations carried out on magnesite refractories made from graded magnesite with small additions of alumina and chromite are described. As a result, a magnesite refractory was produced with good physical properties withstanding more than 50 spalling cycles. Tests in the roof of a small electric arc furnace were successful. Recommendations for the manufacture of this type of refractory bricks on an industrial scale are given.—V. G.

Refractory Materials. J. Apraiz. (*Bol. Min. e Indust.*, 1952, **31,** Nov., 437–443). [In Spanish]. Equilibrium diagrams of interest in the production of refractory bricks are presented and explained.—R. A. R.

Economise in Refractory Materials. A. Macku. (*Hutnik.* (Prague), 1953, **3,** 3, 64–69). [In Czech]. The most efficient use of refractory materials, particularly in steelmaking is discussed, and methods of re-using refractories, such as bricks from dismantled open-hearth roofs, for similar or other purposes are considered. The properties of the ten commonest refractory materials are given in a table.—P. F.

Comparison of Various Linings of Hot Blast Pipes. H. Stempel. (*Demag News*, 1952, No. 129, 25–27). Two linings are compared—one of 150 mm. fireclay and 150 mm. ordinary insulating brick and the other of 80 mm. ordinary insulating brick and 145 mm. of hard-burned, toothed, abrasion-resisting insulating brick. A mathematical analysis is made of the heat losses in the two cases under standard operating conditions. The hard-burned brick lining is shown to be the most efficient and results in a saving of heat and weight.—B. G. B

BLAST-FURNACE PRACTICE AND PRODUCTION OF PIG IRON

Conservation in Production of Iron and Steel. C. Williams. (*Proc. United Nations Sci. Conf. on the Conservation and Utilisation of Resources*, Lake Success, 17 Aug. to 6 Sept. 1949, 2, *Mineral Resources*, 172–176). Sintering and sizing of ore was found to conserve coke and increase output of blast-furnaces; similar results are attained by blast-furnace operation under high top-gas pressure. Oxygen enrichment of the blast is under large-scale investigation and definite information as to its practicability should be forthcoming soon. High ore and iron charges and the use of oxygen for decarburization of the bath are now widely accepted in open-hearth operations where also oxygen and sinter are replacing feed ore.—J. C. B.

Some Aspects of Merchant Iron Furnacing. A. J. Macdonald. (*Blast Furn. Steel Plant*, 1953, **41**, June, 646–650, 655). A detailed account of the blast-furnace practice at the Hanna Furnace Corporation, U.S.A., is given. The plant consists of four blast-furnaces and it can produce almost any grade of iron required.—B. G. B.

The Ironmaking Plant at John Summers and Sons (Shotton Works). J. F. R. Jones and A. Walker. (*J. Iron Steel Inst.*, 1953, **175**, Nov., 313–321). [This issue].

Blast-Furnace Plant at Hawarden Bridge Steelworks. (*Engineering*, 1953, **175**, May 22, 652; June 26, 812). A description, with a detail section drawing and illustrations, is given of the new 1000-ton per day blast-furnace plant of John Summers and Sons, Ltd. The hearth diameter is 27 ft., the furnace volume is 41,370 cu. ft., and the height, including top gear, is 247 ft. 3 in. The plant has been designed as a two-furnace installation, and the second furnace is under construction.—R. A. R.

Kaiser Steel Production Record Set by Coke and Blast-Furnace Practice. W. C. Rueckel. (*J. Met.*, 1953, **5**, Apr., 509–514). The author discusses the evolution of a blast-furnace practice at the Fontana works of Kaiser Steel Corp., which is highly successful in spite of the use of coal which until recently was felt to have poor coking qualities. Research to improve these coking qualities is outlined, and the present practice at the blast-furnace, coke-oven, ore-sizing, and sinter plants is described.—G. F.

Modification in the Design of Blast-Furnace Throats Fitted with Stahler Type Charging Bins. P. Thierry. (*Mét. Constr. Mécan.*, 1953, **85**, Apr., 251–257). The changes in the design of the throat of blast-furnaces which are fitted with 'Stahler' charging mechanism are discussed. The system as now used in France consists of a single bell with a distributor cone underneath. The side walls of the stack are protected. Rapid repair of the throat of the furnace is possible with this headgear.—B. G. B.

Hot-Blast Slide Valves with Low Cost Equipment. W. Dehne. (*Stahl u. Eisen*, 1953, **73**, June 4, 790–791). A hot-blast slide valve assembly is recommended. The sealing rings are made of copper and the valve tongue is in the form of a spiral tube; no alterations to conventional valve seatings are necessary. The cost of construction is low and heat losses and water consumption are reduced.—J. P.

Relation between Blowing Pressure and Percentage Agglomerates in the Völklingen Blast-Furnaces. M. Heczko. (*Centre. Doc. Sider. Circ. Inform. Tech.*, 1952, No. 9, 1263–1270). Brief details of the Dwight-Lloyd sinter plant, which is operated to produce a sinter of minimum ferrous oxide content are given. Tables for three blast-furnaces give daily coke rates, flue dust production, percentage of sinter in burden, and volume and rate of blast. Even with 70% agglomerate, blast pressure does not increase, the only disadvantage being the high top-gas temperature. The porous, little fused sinter requires a burden of well controlled moisture and air infiltration must be kept at a minimum.—A. G.

The Pressure and Supply of Wind to the Blast-Furnace. P. Thierry. (*Mét. Constr. Mécan.*, 1953, **85**, Jan., 15–19). The effect of the amount of wind blown on blast-furnace performance is discussed. The influence of bosh tuyeres, the permeability of the stock column, and the temperature of the blast are three factors which determine the pressure drop across the furnace. The effect of these are considered.

Silicon Content of Blast-Furnace Iron Lowered by Oxygen Injection in Ladle. P. S. Leroy. (*J. Met.*, 1953, **5**, Mar., 396–400). French research work on desiliconization of blast-furnace iron by oxygen injection in the ladle is described. Vertical lances are found to be more efficient than inclined ones, and lancing the ladle during filling gives a more homogeneous metal with smaller metal losses. High-pressure oxygen (about 200 lb./sq. in.) through a non-immersed lance is also advantageous in increasing the efficiency.—G. F.

The Desulphurization of Pig Iron with Acid Slags. H. G. Erne. (*Eidgenössische technische Hochschule*, Zürich, 1952, Promotionsarbeit No. 1995). In experiments carried out by the author siliconized pig iron high in sulphur was desulphurized at temperatures above 1600° C. Silicon and carbon increased sulphur activity and sulphur increased the carbon activity. Desulphurization *in vacuo* was investigated. Depending on the silicon content, acid desulphurization can amount to more than 60% at 1700° C. The most important practical results are: (1) Desulphurization of pig iron depends on its oxygen content; and (2) pig iron can be desulphurized *in vacuo* at 1400° C. in extremely short periods of time.

The Introduction of Powdered Material into the Blast-Furnace Tuyeres. E. Diepschlag. (*Met. u. Giesserei Techn.*, 1953, **3**, Feb., 52–53). Suggestions have been made to desulphurize iron in the blast-furnace by introducing lime into the tuyeres. The author describes an arrangement by which it is possible to prevent an excess of lime in the slag, and to control and adjust the composition of slag in the hearth.—L. J. L.

The Charging of Pulverized Materials into the Blast-Furnace. P. Thierry. (*Mét. Constr. Mécan.*, 1953, **85**, May, 345–351). The charging of very fine material into the blast-furnace is discussed. Injection through the tuyeres is not considered to be very satisfactory. Briquetting the fine material for direct charging into the furnace is the most satisfactory method of treatment.—B. G. B.

Injection into the Blast-Furnace of Solid, Liquid, or Gaseous Materials by Various Methods. M. A. Rygaerts and M. V. Remond. (*Rev. Tech. Luxembourg*, 1953, **45**, Jan.–Mar., 30–42). Many methods of injecting materials into the blast-furnace by the tuyeres have been studied. Solid substances were injected to increase the fluidity of the slag. Fine coke and coal dust have also been injected (the former without success, but the latter gave a top gas of high calorific value). The results of paraffin and fuel oil injections are described, the former was successfully used to clear scaffolds around the tuyeres. Water injection has been employed to remove large scaffolds. Water vapour and oxygen additions to the normal air blast and the injection of blast furnace and coke-oven gas are also discussed.—B. G. B.

Results of an Investigation on Blast-Furnaces with Different Hearth and Hearth Bottom Linings. B. Weilandt. (*Stahl u. Eisen*, 1953, **73**, May 21, 696–704). The properties of fireclay, quartz-fireclay and carbon bricks, concrete, and rammed carbon for furnace bottoms are discussed. Since the heating up of a carbon-lined hearth requires special care, suitable measures are reported. Tar-coke mixtures have proved suitable for patching rammed carbon linings. The influence of a salamander on the life of a furnace bottom was investigated. The temperatures reached at various levels in a new bottom were measured and graphical methods for predicting these temperatures are presented.—J. P.

The Determination of the Gas Composition in the Stack of a Blast-Furnace. P. Papier. (*Mét. Constr. Mécan.*, 1953, **85**, Feb., 89–91). A description is given of a water-cooled probe fitted with a drill for the penetration of the burden of a blast furnace in order to take gas and solid samples across the stock column. Some results obtained are given. A 1·5-kW. motor is used to drive the drill and the time taken to penetrate to the centre is 45 min. The life of a probe is two to three years.—B. G. B.

On the Viscosities of Molten Slags II Viscosities of CaO-SiO_2-Al_2O_3-MgO Slags. Y. Kawai. (*Sci. Rep. Res. Inst. Tôhuku Univ.*, 1952, **A4**, Dec., 615–621). [In English]. A rotating-cylinder viscometer was used with synthetic blast-furnace slags. Diagrams show iso-viscosity lines at 1500° and 1600° C. MgO decreases the viscosity of CaO–SiO_2 slags (within limits) and CaO–SiO_2–Al_2O_3 slags; the effect is marked when SiO_2 and Al_2O_3 contents are reduced. Earlier results of McCaffery and others are compared. MgO has approximately the same effect as CaO.—K. E. J.

Modern Disintegrator Gas-Cleaning. G. Baumann. (*Iron Steel Inst. Translation Series*, 1953, Sept. No. 472). This is an English translation of a paper which appeared in Stahl u. Eisen, 1953, **73**, Apr. 9, 492–494 (*see* J. Iron Steel Inst., 1953, **174**, Aug. 395).

Cleaning Blast-Furnace Gas. S. P. Kinney. (*Brit. Steelmaker*, 1953, **19**, July, 432–437). **New Blast-Furnace Gas Cleaning Facilities at Donora.** S. P. Kinney. (*Iron Steel Eng.*, 1953, **30**, May, 139–142: *Blast Furn. Steel Plant*, 1953, **41**, July, 776–779, 817). The author describes a new cleaning plant recently installed at United States Steel Corp. which eliminates pollution of the atmosphere and waste water by blast-furnace operation. High pressure stationary disintegrator washers and Dorr thickeners are described as well as a primary and secondary Kinney type gas washer.

Electrostatic Precipitators. H. S. Westh. (*Ingenieren*, 1949, **58**, 1, 36–48). Principles of the removal of dust and

fluid suspensions from gases are discussed. A description of various types of electrostatic precipitators, their design, operation, performance, and a comparison with mechanical dust separators are given.—v. g.

Production of Pig Iron in Electric Furnaces. M. O. Sem. (*Inst. Hierro Acero*, 1953, **6,** Jan.–Mar., 16–20). [In Spanish]. The author gives a general account of the Tysland-Hole furnace and discusses its future possibilities. A general review of the Elkem rotary crucible furnace used in the calcium carbide and ferro-alloys industries is also given.—R. S.

International Research on the Low Shaft Furnace. M. Malcor. (*Rev. Tech. Luxembourg.*, 1953, **45,** Jan.–Mar., 1–8). After reviewing the economic conditions which have led the ironmaking industry to consider replacing the normal blast-furnace with the low shaft furnace, the author discusses the advantages of the latter. The low shaft furnace and ancillary plant at Liége to produce 60 to 100 tons of iron per day together with gas of high calorific value are described.

The Formation of a Professional Body of Blast-Furnace Personnel. P. Thierry. (*Mét. Constr. Mécan.*, 1953, **85,** Feb., 92).

TREATMENT AND USE OF SLAGS

Developments in the Production of Slag Wool in the Vereinigte österreichische Eisen- und Stahlwerke, Linz. O. Rosenauer. (*Stahl u. Eisen*, 1953, **73,** June 18, 845–847). The use of coke-oven gas and an electric furnace for slag melting has made it possible to obtain correct temperatures and to adjust compositions by the addition of silica, thus improving the quality and the yield of slag wool. The slag is sprayed through a nozzle and the wool removed on a rotating drum in the form of a continuous band. In another method, the drum is replaced by an endless belt carrying individual boxes so that the slag wool is preformed into mats of desired size. The latest development involves the addition of small quantities of an emulsion at the spray nozzle to coat the wool fibres with a thin layer of water repellant.—J. P.

The Production of Slag Ballast. C. R. Bennett. (*Rail Steel Topics*, 1952, **1,** 4, 4–9). The author describes the production of railway ballast from blast-furnace slag. The operations involve selection of a suitable composition, quick cooling, crushing, and screening.—A. D. H.

Expanded Blast Furnaces Slag for Use as Light Weight Concrete Aggregate. R. W. Miller. (Amer. Iron Steel Inst. : *Blast Furn. Steel Plant*, 1953, **41,** June, 635–638, 645). Methods of producing expanded and granulated blast-furnace slag for use as light weight concrete aggregate are described.

Manganese from Open-Hearth Slag Could Supply Half of U.S. Needs. R. C. Buehl, M. B. Royer, and J. P. Riott. (*J. Met.*, 1953, **5,** Apr., 520–521). The U.S. Bureau of Mines has developed a two-stage cyclic process for producing a slag with 55–60% Mn from open-hearth slag. This article describes further pilot-plant work on a larger scale in which a 1-ton side-blown converter was used to establish the cyclic converter operation. It is estimated that half the manganese requirements of the United States could be supplied by this process.

DIRECT PROCESSES

The Production of High Purity Iron by Direct Reduction from the Ore. J. E. Ferry. (*Australasian Eng.*, 1952, Nov. 7, 70–73 ; Dec. 8, 48–52). Details are given of the Duffield method of smelting, in which only sufficient carbon for the reduction of the iron oxides exists, in the presence of a oxidizing atmosphere. The high-purity, high-melting-point (1500° C.) iron, is separated at 1450° C. from a low-melting-point slag and transferred to an electric furnace for melting and degasification. Details of the iron conglomerate, and the heat balance and heat generated are given. The vast ore reserves in Oxfordshire and Northamptonshire, in the form of hydrated oxides are well suited to the Duffield process. The iron-rich residues from pyrites used in the chemical industry can also be used to produce iron of high purity. This raw material is practically valueless for standard smelting methods because of the sulphur content. Further details of the economics of the Duffield method and the use of Duffield iron as a base for alloy steel production are given.—P. M. C.

Sponge Iron. P. E. Cavanagh. (*Metal Progress*, 1953, **63,** Mar., 67–69). The commercial production, in Sweden and Mexico, of sponge iron for melting stock in steelmaking is about 50,000 tons per year. The tunnel kiln plant in Mexico and the Wiberg-Söderfors furnaces in Sweden are considered and the costs and performances are compared.—B. G. B.

Sponge Iron Used as a Precipitant for Copper. F. F. Frick. (*J. Met.*, 1953, **5,** Mar., 395). The Anaconda Copper Mining Co., Montana, uses sponge iron as a precipitant for copper in solution resulting from desliming the ore in dilute sulphuric acid. Brief details are given of pilot plant trials on the production of a satisfactory sponge iron at low cost.—G. F.

PRODUCTION OF STEEL

A Special Survey of the Shotton Works of John Summers and Sons, Limited. (*Brit. Steelmaker*, 1953, **19,** May, 231–340). A very compressive description is given of the iron and steel works, including the recently completed extensions, of John Summers and Sons, Limited, Shotton.—R. A. R.

Steel Expands in the Detroit Area. R. Sergeson, H. L. Klinker, and A. Krause. (*Iron Steel Eng.*, 1950, **30,** May, 114–118). This article, in three parts, describes the expansion at The Rotary Electric Steel Co., the Great Lakes Steel Corp. and Ford Motor Co., giving particulars of production and some of the new equipment.—M. D. J. B.

Strutting a Steelworks Building. W. Möller. (*Demag News*, 1952, No. 129, 13–15). A steel plant extension project involving the installation of a new open-hearth furnace, which was carried out in the minimum of time without interfering with the usual steel making operations, is described.—B. G. B.

The Iron Works Combine Nowa Huta. Schweda. (*Met. u. Giesserei Techn.*, 1952, **2,** Sept., 323–325). A description is given of the integrated iron and steelworks, construction of which has commenced at Nowa Huta, near Cracow.—R. A. R.

Observations on the Construction of Modern Steel Works. H. Stollberg. (*Met. u. Giesserei Techn.*, 1952, **2,** Aug., 258–262). The improvements advocated for the construction of steelworks in Eastern Germany include a continuous scrap supply, increase in fuel gas production, increased efficiency of open-hearth furnaces, better quality refractory building materials, fully automatic control, and better measuring instruments.—L. J. L.

Note on Converter Bottoms Made with Metal Tubes. A. Richard. (*Centre Doc. Sider. Circ. Inform. Tech.*, 1952, No. 8, 1124–1129). Trials on converters with metal blowing tubes are described. Advantages achieved are less blocking, an increased bottom life, less maintenance during use, and a decrease in blowing time due to reduced iron losses. The only difficulty encountered was in removing the tube bottom before it was completely worn out.—A. G.

The Importance of the Basic-Bessemer Process in Metallurgy. F. Franz. (*Met. u. Giesseri Techn.*, 1952, **2,** Oct., 345–347). The economies of the basic-Bessemer and open-hearth processes are compared, and the use of oxygen-enriched air is advocated to reduce the nitrogen content of basic-Bessemer steel.—R. A. R.

The Heating of Open-Hearth Furnaces with Fuel Oil. J. E. Lafon. (*Mét. Constr. Mécan.*, 1953, **85,** Mar., 205–209 ; Apr., 295–299). The results obtained with fuel oil in the United States and Great Britain for heating open-hearth furnaces are discussed with particular reference to increased application in France.—B. G. B.

Use of Coke Oven Tar for Heating Open-Hearth Furnaces. M. Hartemann. (*Centre Doc. Sider., Circ. Inform. Tech.*, 1952, No. 7, 976–982). An account of the successful replacement of high sulphur fuel oil by dehydrated coke-oven tar is given. No modifications to burners or atomizing fluid were made. Heat consumption per ingot-ton was slightly lower when using tar but consumption of silica bricks increased 25%. However, total difference in construction costs was slight and the reduction in fuel costs was 14%. Provided high quality refractories are employed, tar gives a remarkable saving in melting time and good desulphurization of steels (average sulphur in finished steel 0·025% compared to 0·042% for fuel oil).—A. G.

The "H.P." Burner for Open-Hearth Furnaces. (*Radex Rundschau*, 1951, No. 8, Dec., 361–362). A new system of burner construction invented at the Radenthein magnesite works is described in brief, a life increase of 30% being claimed in comparison with customary systems.—E. C.

Flow Pattern Studies. J. H. Chesters. (*Rail Steel Topics*, 1952, **1,** 3, 4–11). An account is given of flow pattern studies in open-hearth furnaces and locomotive fire boxes. The way

in which studies of similar problems should be undertaken is discussed.—A. D. H.

The Automatic Regulation of Open-Hearth Furnaces. Z. Rychlik. (*Hutnik* (Katowice), 1953, **20,** 2, 56–63). [In Polish]. The performance of an open-hearth furnace and the measurements necessary for its assessment are discussed. The automatic regulation of : (*a*) Pressure in the working space of the furnace ; (*b*) temperature of the furnace roof ; (*c*) change of the direction of flow of gases ; and (*d*) the combustion process are described.—V. G.

Vanadium in the Acid Steelmaking Process and Its Deoxidizing Capacity. I. A. Popov and B. W. Stórk. (*Met. u. Giesserei Techn.*, 1952, **2,** Oct., 362–364). In slags consisting of the oxides of iron, silicon, manganese and vanadium, the last constitutes a phase corresponding to the iron-vanadium spinel $FeO.V_2O_3$.—L. J. L.

Degree of Oxidation of Bath and Slag in the Basic Open-Hearth Furnace. M. Bettembourg. (*Centre. Doc. Sider. Circ., Inform. Tech.*, 1952, No. 9, 1271–1281). Bath oxidation is considered in relation to the establishment of a stationary state in which all the oxygen transferred from flame to slag is utilized in decarburization. Rate controlling factors are discussed and a review of various authors' values of [C] [O] as a function of [C] is given. A definite correlation between ferrous oxide in slag and metal exists, the ratio varying with basicity. Factors governing the oxidizing power of the flame and intervention by the operator are considered.—A. G.

Effect of Chromium on the Thermodynamic Activity of Carbon in Liquid Iron. F. D. Richardson and W. E. Dennis. (*J. Iron Steel Inst.*, 1953, **175,** Nov., 257–263). [This issue].

The Equilibrium Controlling the Decarburization of Iron–Chromium–Carbon Melts. W. E. Dennis and F. D. Richardson. (*J. Iron Steel Inst.*, 1953, **175,** Nov., 264–266). [This issue].

Metallic Oxidation in Chromium Steel Melting. D. C. Hilty, G. W. Healy, and W. Crafts. (*Trans. Amer. Inst. Min. Met., Eng.*, 1953, **197:** *J. Met.*, 1953, **5,** May, 649–653). From a theoretical extension of the Cr–C temperature relationship in chromium steels to low chromium contents, and by correlating the chromium/iron ratios in slag and metal, the authors have developed a method for estimating the amount of metallic oxidation during the oxidizing period of a chromium steel heat. Application of the method indicates that, due to temperature limitations, metallic oxidation may be excessive for very low carbon steels charged with more than a small amount of chromium—G. F.

Refining a High-Phosphorus Iron with Oxygen in the Open-Hearth Furnace. G. Husson and P. Bettembourg. (*Rév. Mét.*, 1953, **50,** May, 357–371). Trials on six casts in two groups are reported in which phosphoric irons were treated with an oxygen lance in the open-hearth furnace. After a consideration of material balances and the metallurgical processes involved, a charging and blowing procedure is put forward. A production rate with hot iron equal to that normally obtained with scrap is reported with a heat saving of 22·5%. However, the cost of oxygen results in an increase in the cost of steel of 1700 francs/ton. The fumes problem is considered. The method, by permitting the use of low phosphorus iron in the open-hearth furnace, could ease the scrap situation.—A. G.

Chemical Behaviour of Sulphur in Iron and Steelmaking. J. Chipman. (*Metal Progress*, 1952, **62,** Dec., 97–107). The author discusses earlier papers on the partition of sulphur between slag and metal in the open-hearth furnace and the blast-furnace. Measurement of the activity of sulphur in the bath and of the desulphurizing power of slags in relation to their composition is also discussed. The results obtained indicate that the control of sulphur depends greatly on the control of oxygen, and that more complete and more rapid desulphurization depends on : (1) Basicity of the slag, (2) the effectiveness of slag-metal contact, and (3) the oxygen potential.—B. G. B.

The Problem of Desulphurizing Steel. J. Trapl. (*Hutnik*, (Prague), 1953, **3,** 3, 59–63). [In Czech]. The chemistry of the removal of sulphur in steelmaking is reviewed. The influence of the lime in the slag on the partition coefficient of sulphur in the slag-metal system during the oxidizing period is considered, and the damaging effects of sulphur and phosphorus on the impact strength and other properties of steels discussed. It is shown that the views of Fohs, Ossan, and Schleicher, according to whom the beneficial influence of fluoride additions to the bath results from the elimination of some of the sulphur in the form of volatile fluorides, is erroneous. The influence of fluoride is ascribed to the increase in the molar concentration of anions in the slag, caused by the addition, and by the decrease in its viscosity and increased chemical activity. The effects of various elements and compounds on desulphurization are also dealt with in the light of the ionic theory.—P. F.

The Development of Efficient Tilting Open-Hearth Furnaces Fired with Coke-Oven Gas. H. Trenkler. (*Radex Rundschau*, 1952, No. 2, Apr., 84–92). A 200-ton tilting open-hearth furnace of the Vereinigte Oesterreichische Eisen und Stahlwerke at Linz was rebuilt in the post-war period to adapt it to coke-oven-gas firing. Details of this reconstruction are given together with information on subsequent experiences in operation, further modifications and improvements. Data for six campaigns are tabulated and discussed.—E. C.

Physically Hot Iron Preferred for Open Hearths. D. M. Morrison. (*J. Met.*, 1953, **5,** Apr., 522–524). In view of open-hearth operators' preference for physically hot iron, the author discusses blast-furnace practice which usually favours a lean slag resulting in a relatively cold iron. From available data and personal observations he concludes that the optimum slag condition is neither exessively lean nor limy, and depends largely on time, place, and materials available.—G. F.

Production of Draught in Industrial Furnaces, with Particular Reference to the Open-Hearth Furnace. M. Hansen. (*Stahl u. Eisen*, 1953, **73,** June 18, 838–844). The amount of draught required to remove waste gases from a variety of open-hearth fuels is discussed. The draught may be induced naturally or artificially. Natural draught is produced by tall chimneys, and nomograms are given for determining the draught as a function of self-losses, diameters, siting of chimney and atmospheric conditions. The use of fans or blowers for artificial draught production is described, reference being made to the Morgan-Isley method used in the U.S.A. and to the use of ejectors in Germany.—J. P.

The Behaviour of Hydrogen in Steelmaking. E. Piper, H. Hagedorn, and H. Backes. (*Stahl u. Eisen*, 1953, **73,** June 18, 817–825). A new method is described for the determination of the hydrogen content of the metal at all stages from pig iron to finished ingot. Moisture, in the charge and additions to the steel furnace, has no influence on the hydrogen content of steel in the furnace providing that this is covered with a slag blanket and that water does not come in direct contact with the uncovered steel. Exceedingly little water vapour in the heating gases is transferred to the steel through the slag. In the basic open-hearth and basic electric furnaces, an increase in hydrogen is noted from the middle of the decarburizing period. The hydrogen pick-up is greatest in the basic open-hearth furnace after the end of the carbon period and in the electric furnace after the removal of the black slag. Attempts have been made to relate hydrogen pick-up to interactions between bath, slag and water vapour. In the open-hearth furnace, hydrogen pick-up occurs when the ferric oxide in the slag increases. There is no hydrogen pick-up in the acid high-frequency furnace. With all melting procedures, the hydrogen content of the steel generally increases after tap, the extent of the increase depending on the tapping and teeming procedure and the type of mould employed.—J. P.

The Different Mechanisms of the Action of Manganese During Steel Refining. M. Bettembourg. (*Centre. Doc. Sider. Circ. Inform. Tech.*, 1952, No. 7, 983–990). The mechanism of manganese dioxidation by reaction with ferrous oxide to give insoluble manganese oxide is considered to be only partially correct. Theoretical relationships between manganese and oxides and sulphides are discussed together with the practical effects of manganese on forgeability and the behaviour of steels.—A. G.

Stachanovite Method of Carrying out Rapid Melts. A. Děkanovský. (*Hutnik*, (Prague), 1953, **3,** 4, 77–79). [In Czech]. Methods of reducing the heat time of a 200-ton fixed basic open-hearth furnace from 12 to about 7 hr. are described. The procedure for charging scrap and hot metal is explained, and air and fuel rates during repairs, charging, pouring, melting, and finishing are given.

Soviet Methods and Results of High Speed Melting in Open-Hearth Furnaces. K.-F. Lüdemann. (*Met. u. Giesserei*

Techn., 1952, **2**, Oct., 338–344). The author describes the construction and equipment of open-hearth furnaces in Russia and compares Venturi and Maerz ports and points out peculiarities in Russian smelting practice.—L. J. L.

Increasing the Productivity of Open-Hearth Furnaces. J. E. Safon. (*Mét. Constr. Mécan.*, 1953, **85**, May, 393–397). Methods of decreasing the time of the five operations, namely, fettling, charging, melting, refining, and tapping are discussed.—B. G. B.

The Basic Lining of Slag Pockets and Checker Chambers of Open-Hearth Furnaces. L. Hütter. (*Radex Rundschau*, 1951, No. 7, Nov., 307–321). The all-basic construction of open-hearth furnaces is shown to be firmly established and its development completed. Experience in operation is described and discussed with regard to service life and performance. Improvements of weak parts of the furnace below stage level are suggested. The advantages of using Radex bricks for slag pockets and checkers are stressed, and results of experiments and calculations are quoted in support of these claims.

Metallurgical Suitability of Open-Hearth Furnaces of the Maerz Type. W. Titze. (*Radex Rundschau*, 1951, No. 6, Oct , 234–242). Metallurgical and operating conditions of a Maerz type open-hearth furnace were investigated and compared with corresponding data for conventional types. Various practices and steel qualities are considered and it is claimed that the Maerz furnace permits a stronger and more easily controllable oxidizing rate which makes it suitable for the production of high grade steels.—E. C.

Mechanical Fettling of Open-Hearth Furnaces with Dolomite. (*Biuletyn Informacyjny Ministerstwa Hutnictwa*, 1953, No. 3, 9–11 : *Hutnik* (Katowice), 1953, **20**, 3). [In Polish]. The description of a mechanical fettling machine of Polish construction is given. The acceleration is imparted to a stream of dolomite by a rotating drum with blades and an endless belt running around the periphery of the drum.—V. G.

Hot Repairs to Ladles. (*Hutník*, (Prague), 1953, **3**, 3, 69–70). [In Czech]. A method of repairing 3-ton ladles without allowing them to cool is described. By this method a ladle can be maintained in continuous use for 100 to 120 heats.—P. F.

Blasted Bales Give Up Secrets. (*Steel*, 1953, **132**, June 8, 97–98). To deal with increasing purchases of baled scrap, Bethlehem Steel Co. have developed a system of inspection in which sample bundles are blasted open with dynamite for examination.—D. L. C. P.

Atlantic Steel Using Cold Metal Charge Finds Costs in Favor of Electric Furnace. J. E. Wilbanks. (*J. Met.*, 1953, **5**, May, Section 1, 627). The 80-ton open-hearth furnaces and 75-ton electric furnace at the Atlantic Steel Co. produce the same grades of steel by cold-metal practice. Overall costs have been compared over six months. and found to be in favour of the electric furnace.—G. F.

Rimming Steel Production Rate Studied by Multiple Correlation. W. W. Black and C. R. Taylor. (*J. Met.*, 1953, **5**, Mar., 401–403). Data from 100 rimming steel heats made in a 70-ton electric furnace have been collected, and the effect of several variables on the production rate has been studied. It is found that charging times, delays, lime additions, and proportion of bundles in the scrap shou'd be kept to a minimum, and, in triple charging, oxygen is beneficial only in the second and third charges.—G. F.

Electric Ore Smelting Passes Tests. H. S. Newhall. (*Steel*, 1953, **132**, May 11, 144–147). The author surveys achievements and possibilities in smelting and refining in electric furnaces and concludes that possilibities of making steel from ore in the arc furnace grow as experimental data points to operating advantages.—D. L. C. P.

Use of Rare-Earth Metals and Compounds in Stainless Steel Melting. C. B. Post and H. O. Beaver. (Amer. Iron Steel Inst. : *Blast Furn. Steel Plant*, 1953, **41**, June, 627–634, 645). Misch-metal addition to high-alloy austenitic stainless steels, which are inherently hot-short, can transform them into ductile alloys. Additions to low-alloy austenitic steels, which are inherently ductile, can lead to improvements in hot workability. The mechanism of the above changes is uncertain. Additions of rare earth oxides were found to lead to no significant improvement in the steel and no trace of rare-earth metal was retained in the steel.—B. G. B.

Refining Corrosion-Resistant Steels with Oxygen in the Arc Furnace. F. Cless (*Redax Rundschau*, 1951, No. 8, Dec., 341–354). The importance of oxygen refining is emphasized and physico-chemical principles and the technique of the process are described. Factors influencing quality and economics are evaluated graphically and refractory problems are discussed.—E. C.

The Use of Gaseous Oxygen in Electric Steel Making. R. B. Halsey. (Australian Inst. Metals : *Australasian Eng.*, 1953, Jan. 7, 66–69). The development of oxygen blowing for the refining of electric steel in a 15-ton, basic lined, 3500-kVA. Héroult type furnace at the Newcastle works of the Commonwealth Steel Co. is described. The general application of gaseous oxygen is briefly reviewed, and the technique established in treating the particular case of 18/8 stainless steel is described. (12 references).—P. M. C.

Chemical Processes in the Electric Are Furnace. A. Lincke. (*Met. u. Giesserei Techn.*, 1953, **3**, Apr., 152–153). The behaviour of carbon, manganese, silicon, phosphorus and sulphur with special references to the difference between the reactions in the electric arc and open-hearth furnaces, is made.—L. J. L.

Pre-Refining Processes Speed up Steelmaking: Rocking Resistor Furnace for Desulphurization. C. E. A. Shanahan and F. J. Lund. (*Iron Coal Trades Rev.*, 1953, **166**, Mar. 27, 701–705). The authors discuss the latest techniques in prerefining processes which are beneficial in reducing metallurgical load. An experimental rocking resistor furnace has been designed for desulphurization by agitating slag and metal, and results obtained using this furnace show that rocking increases the rate of desulphurization.—G. F.

The Basic Lining of High Frequency Induction Furnaces. F. Sellner. (*Radex Rundschau*, 1951, No. 8, Dec., 355–360). The importance, use, and development of high-frequency induction furnaces are described. Basic linings are advocated, and metallurgical and economic advantages are claimed in comparison with acid linings.—E. C.

Thermodynamic Study of Dilute Solutions of Carbon in Molten Iron. F. D. Richardson and W. E. Dennis. (*Trans. Faraday Soc.*, 1953, **49**, Feb., 171–180). The equilibria between CO + CO_2 gas mixture, and carbon dissolved in molten iron were investigated at 1560°, 1660° and 1760° C. for ratios of CO/CO_2 up to 800/1. A bead of liquid alloy was brought into equilibrium with a gas stream of known composition, quenched, and analysed for carbon. The activities of graphite and iron are plotted as a function of composition.

Steel Ingot Technique : I. Introduction; II. Ingot-Mould Design; III. Ingot Feeder Heads; IV. Ingot Surface Defects; V. Internal Structure of Ingots; VI. Casting-Pit Ancillary Equipment; VII. Liquid Steel for Ingot Production; VIII. Casting-Pit Procedure and Teeming Technique; IX. Casting-Pit Refractories; X. Engineering Aspects of Casting-Pit Layout; XI. Future Developments in Steel Casting. H. O. Howson. (*Iron Coal Trades Rev.*, 152, **165**, Nov. 28, 1169–1178 ; Dec. 5, 1233–1240 ; Dec. 12, 1283–1290 ; Dec. 19, 1353–1362 ; Dec. 26, 1391–1398 ; 1953, **166**, Jan. 9, 85–93 ; Jan. 16, 131–138 ; Jan. 23, 205–208 ; Jan. 30, 253–262 ; Feb. 6, 313–317 ; Feb. 13, 373–379 ; Feb. 20, 425–430 ; Mar. 6, 535–541 ; Mar. 13, 605–609, 611 ; Mar. 20, 661–666 ; Mar. 27, 717–722 ; Apr. 3, 781–787 ; Apr. 10, 829–833 ; Apr. 17, 890–892 ; Apr. 24, 923–928 ; May 1, 997–1004 ; May 15, 1107–1114 ; May 22, 1173–1179). Following a brief history of commercial ingot production, the author discusses in Part 2 the design and service life of ingot moulds, outlining the design recommendations of the B.I.S.R.A. Ingot Moulds Sub-Committee. Cracking, crazing, and the mould/ingot weight ratio are considered in relation to design and consumption. In Part 3 are classified the types of feeder head acceptable in modern practice. Recommendations for heads for different mould types are given, and special refractory shapes such as the recessed tile, developed for improved efficiency, are discussed. Ingot surface defects are discussed in Part 4, particular attention being given to cracks, processing defects, and double skin. Investigations of factors influencing ingot cracking are summarized, followed by a discussion of blowhole formation and their effect during subsequent processing. In Part 5, the internal structures of ingots ranging from 'wild' to fully-killed are briefly described, and attention is given to blowhole distribution, piping, axial unsoundness, and segregation. The influence of there factors on the amount of discard is considered. The effect of casting method and mould design on segregate distribution is also discussed. The casting-pit

ancillary equipment described in Part 6 includes mould bottom-plates for direct and bottom teeming, splash tins, tundishes, troughs and downrunners, and the design and construction of ladles are discussed. Part 7 deals with the factors affecting the preparation of steel for casting good ingots, including steel analysis, state of oxidation, deoxidation products and non-metallic inclusions, liquid steel temperature, and the presence of hydrogen and nitrogen. The control and measurement of bath and ladle temperatures are also discussed. Part 8 gives practical information on casting-pit procedure and teeming technique. The choice of direct or bottom teeming and the appropriate mould arrangements are considered. Teeming rates, particularly of rimming steels, are discussed and the selection of type and size of nozzle is considered. The required properties of casting-pit refractories are dealt with in Part 9, and factors influencing the design of nozzles, stoppers, runner tubes and trumpets are outlined. Casting-pit layout and the problems of handling large masses of molten metal are discussed in Part 10. The fundamentals of casting-pit layout are outlined, and other features described are ladle-car design, crane distribution, ingot stripping and handling methods, slag disposal, and pit layout in different types of plant. In Part 11 possible future trends are considered, and continuous casting is briefly reviewed. Factors influencing ingot yield are dealt with and information on costs of operation and equipment is given.—G. F.

A Mobile Ingot Mould Grid Helps to Save Space in the Casting Pit. W. Stieg. (*Met. u. Giesserei Techn.*, 1953, **3**, Apr., 130–131). An old railway truck was converted for the reception of 16 to 22 ingot moulds by replacing the floor with a grating and fitting a railing and a platform. The new form of ingot bogie is described.—L. J. L.

Large Silicate Inclusions in Acid Open-Hearth Ball-Bearing Steel During Various Stages of Manufacture. S. Bæckström. (*Jernkontorets Ann*, 1953, **137**, 4, 117–127). [In Swedish]. The author gives results of sample counts of silica inclusions over 10μ in three charges having 1·04%, 1·01% and 1·01% C, and 1·45%, 1·56%, and 1·55% Cr. The number of particles per 10 sq. cm., average diameter and volume percentage were ascertained. Addition of silicomanganese and ferrochrome increased the number of inclusions from 0·2 to 0·5 $\times$ 10^{-3} vol.-%. Immediately after pouring it was 1·5 $\times$ 10^{-3} vol.-%.

Determining Coarse Slag Inclusions in High Carbon Steels. G. Helmer. (*Jernkontorets Ann.*, 1953, **137**, 4, 128–139). [In Swedish]. Two 90-mm. rolled steel samples (0·96% C, 0·39% Cr, acid open-hearth and 0·93% C, 0·34% Cr., acid high-frequency furnaces) were subjected to a step-down turning test. Slag inclusions in the five turned zones were counted at depths up to 25·5 mm. Tabulated results show inclusions larger or smaller than 1·25 mm.—G. G. K.

Causes of Ingot Losses in Cropping. W. Moll. (*Met. u. Giesserei Techn.*, 1952, **2**, July, 226–229). The causes of blistering, piping and fissures in various types of the ingot are discussed.—L. J. L.

PRODUCTION OF FERRO-ALLOYS

Ferro-Alloys. Part I. L. Schmidt and F. Harms. (*Radex Rundschau*, 1952, No. 1, Feb., 20–32). The various fields of application of ferro-alloys are surveyed and methods of production are described and classified. Their applications for reduction refining, and alloying are considered, together with physical and chemical principles for these. Ferrosilicon is dealt with in detail, covering all aspects of production, uses, the most important products, and world requirements. The production and applications of calcium silicide are dealt with in brief.—E. C.

Use and Importance of Manganese in Metallurgy. O. Hájiček. (*Hutník*, (Prague), 1953, **3**, 4, 90–92). [In Czech]. A survey is made of the history of the discovery and manufacture of manganese and its principal alloys, with special reference to their uses in steelmaking. The Brazilian manganese ore deposits are considered of great potential importance.

The Boron Treatment of Iron in the Gas Phase. E. Eipeltauer. (*Metallkundliche Berichte*, 1951, No. 12). Processes for the production of boron-iron alloys are described. Experiments in which iron was treated with BCl_3 at high temperatures have shown : (1) Optimum temperatures are between 950° and 1050° C. ; (2) a chemical reaction takes place leading to FeB formation ; (3) the thickness of the layer is dependent on temperature, duration of exposure, partial pressure of BCl_3 and carbon content of sample ; and (4) the layers adhere firmly to the iron surface and are very hard and brittle. The hardening capacity of boron steels is briefly discussed.—L. J. L.

FOUNDRY PRACTICE

Design, Installation and Operation of a Water-Cooled Cupola. J. W. Dews. (Inst. British Foundrymen : *Foundry Trade J.*, 1953, **94**, May 7, 519–522 ; May 14, 561–565). After describing the situation existing at the foundry before water-cooling of cupolas was envisaged, the author records the reasons which led to its adoption, first experimentally, then on a production basis for a 38-in. dia. cupola running for melts lasting about 4 hr. Experiences are given in applying the same type of water-cooling segments to a pair of new cupolas of 24 in. inside dia. running for 8-hr. melts. In this case, also, operating details and the economies made are given, as well as empiric equations for the heat balance. L. E. W.

Hot Blast Cupolas. A. Upmalis. (*Tekn. Tidskr.*, 1953, **83**, May 12, 407–410). [In Swedish]. The conventional cold blast cupola has an efficiency of only 29%, heat losses being 60%. Americans, by passing outlet gases through an air preheater, reduced their temperature from 935° to 315° C. and raised blast temperature to 280° C. Advantages are : 25% less blast, 50% less pressure, 20–40% less coke and thus less sulphur in the iron. European practice is to have blast at 350–450° C. with outlet gases less than 500° C., giving a ratio of 6–7 kg. coke to 100 kg. iron. Details are given of a typical German unit equipped with a preheater rated at 3 kg. cal./sq. in./hr., giving a metal temperature of 1500° C. Forty German plants have so far been converted to hot blast working.—G. G. K.

Degree of Combustion in the Cupola W. v. Preen. (*Gjuteriet*, 1953, **43**, Feb., 25–28). [In Swedish]. Taking Jungbluth's work as a basis, the author develops equations from a simplified heat balance which permit calculation of the combustion rate as a function of the controlling variables such as coke size and blast temperature. Curves show the relationship between outlet gas temperature, degree of combustion and cupola heat release, also the variation in available heat with blast temperature for different outlet gas temperatures and combustion rates. The method is applicable to both hot and cold blast working.—G. G. K.

Inter-Relation of Combustion and Metallurgical Reactions in the Cupola. D. Fleming. (*Inst. Brit. Foundrymen*, June 16–19, 1953, *Advance Copy No.* 1065). The 'cupola behaviour diagram' of Jungbluth and Korschan is discussed and it is shown that the CO/CO_2 ratio is controlled almost entirely by the iron/carbon ratio of the charge. A reaction temperature is defined and the effects of carbon/iron ratio, coke size, and blast temperature are considered. An account is given of the mechanism of combustion and the reactions occurring in the shaft, melting zone and hearth. Control of a cupola producing soft grey iron could be achieved with a combination of hot-blast, water-cooling, and continuous tapping.—B. C. W.

American Experiences with Basic Cupola Melting. S. F. Carter. (*Inst. Australian Found.*, Victorian Div., 1952 Convention, 119–139). The basic cupola is discussed, and the factors for removal of phosphorus and sulphur are enumerated. The author describes, with detailed examples, production experience in the foundry of the American Cast Iron Pipe Co., refractory maintenance, variation of slag basicity, production of nodular iron and special irons, and appearance of slags of different basicities. General operation and control are discussed. Production experience with water-cooled basic cupolas is also described.—P. M. C.

Air Volume Control in the Cupola. R. C. Davidson. (*Inst. Australian Found.*, Victorian Div., 1952 Convention, 197–208). The importance of air in the cupola is discussed, with a detailed consideration of the reactions during combustion. The method of calculating the air volume for optimum operating conditions, and the adjustment of this basic air volume for variation in atmospheric conditions are described. Numerical examples are given.—P. M. C.

Superheating of Cast Iron by Oxygen Injection into the Forehearth of the Cupola. (*Foundry Practice* (Moscow), 1951, **9**, 15–18 : *Met. u. Giesserei Techn.*, 1953, **3**, Jan., 10–13). Theory and practice of injecting oxygen into liquid iron in the forehearth of the cupola are discussed. From 5 to 6 cu. m. oxygen per ton of iron raises the bath temperature 80°–100° C.,

and 8 to 10 cu. m./ton brings the metal up to 1500°–1520° C. upon which the carbon begins to burn vigorously. With iron initially at 1380° C., the carbon can be reduced by 0·2–0·3% by injecting 7–10 cu. m./ton. The oxygen pressure should not be less than 5–6 atm. and the size of the oxygen nozzles should be from 2–3 mm. for 3 tons/hr. iron production to 4–5 mm. for 8 tons/hr. Data on the slags produced and the advantages of the process are presented.—R. A. R.

Self-Cleaning Twin Tuyeres. (*Fonderie*, 1953, **85,** Feb., 3317–3321). The number of tuyeres on a cupola can be doubled so that they are arranged in a number of pairs, one of each pair blowing while the other is idle. This causes any slag at the nose of the idle tuyere to melt off, so keeping the tuyeres clean. Different designs for such a system are described.—J. MCK.

A New Process for the Production of High Quality Grey Iron. N. N. Roubzov. (*Engineering News* (Moscow), 1948, No. 12, 42–44 : *Met. u. Giesserei Techn.*, 1953, **3,** Jan., 35–36). This article describes the process developed at the Voikov foundry, Moscow, in which large quantities of basic iron are added to the charge in order to obtain grey iron in the cupola free from flakes of undissolved graphite. To modify graphitization, 10–15% of ferrosilicon are added in the cupola.—L. J. L.

The Economic Success of Rationalized Melting in the Cupola. G. Sper. (*Met. u. Giesserei Techn.*, 1952, **2,** Dec., 433–436). The beneficial effect of a change over to G. Mayer's melting process is shown. Coke per charge was reduced to 10% ; 8 kg. of limestone instead of 18 kg. were added to 400 kg. of iron ; desulphurization with soda was discontinued ; volume of blast was 60–65 cu. m./min. ; and blast pressure was 620–640 mm. water gauge.—L. J. L.

Melting in the Cupola without Pig Iron. P. Selack and J. Hellmuth. (*Met. u. Giesserei Techn.*, 1952, **2,** July, 218–221). The standard charge used consisted of 70% external and 30% internal scrap. By keeping acid slag from the iron bath, insisting on basic or neutral ladle linings, and preventing resulphurization by refraining from adding sand to thicken the slag, good results were obtained. The average analysis of the iron obtained was C 3·13%, Si 2·78%, Mn 0·78%, P 0·87%, and S 0·125%.—L. J. L.

The Desulphurization on a Highly Basic Tar Dolomite Lining as an Aid to Melting without Pig Iron. H. Maier. (*Met. u. Giesserei Techn.*, 1952, **2,** Dec., 422–423). A description is given of the basic lining of a forehearth with a tea-pot spout. Even when tar and dolomite iron were mixed in the right proportions, the walls collapsed until the hearth was almost hermetically sealed ; the tar coked after 10 hr. and sintering of the dolomite occurred during the first melt. The total sulphur content was reduced by 50 or 60%.—L. J. L.

The Present State of Spraying Methods for the Repair of Linings in Cupola Furnaces. K. Krämer. (*Met. u. Giesserei*, **40,** Apr. 2, 165–167). The improvement of cupola linings by spraying methods offers the possibility of simplifying and shortening repair work by carrying this out before the furnace is quite cool. The recently developed moistening of the spray with about 4% water has reduced the dust formation and waste without influencing the spraying machine adversely. The effect of the furnace diameter on the economics of the process is discussed.—R. J. W.

Ramming Cupola Linings. G. Sper. (*Giesserei Techn.*, 1952, **2,** Aug., 284–288). Particulars are given of a high-silica ramming material for lining and patching cupolas.—R. A. R.

Measurement of the Actual Blast Volume in a Cupola. S. Pelczarski. (*Przeglad Odlewnictwa*, 1953, **3,** 2, 42–43). [In Polish]. The measurement of blast losses at various blast pressures is described. These determinations should be made regularly since, in general, losses are surprisingly high.—V. G.

Unalloyed Cast Iron—Its Composition and How to Calculate It Having Regard to the Tensile Strength Required. H. Poetter. (*Met. u. Giesserei Techn.*, 1953, **3,** Feb., 45–51). The tensile strength of cast iron depends more on the quantity and formation of the graphite than on the composition and texture of the metal base. The required tensile strength can therefore be attained if due attention is paid to all the factors of importance, especially to superheating, slag treatment, and cooling velocity.—L. J. L.

Graphs in the Foundry Assist Technical Control. J. Pascal. (*Mét. Constr. Mécan.*, 1953, **85,** Mar., 167–173 ; Apr., 259–263 ; May, 357–358). The author shows how the presentation of foundry operating data in graphical form can assist in improving the efficiency of a number of foundry processes.

Fluidity Test for Quality Control in the Foundry. T. P. Yao. (*Foundry Trade J.*, 1953, **94,** May 21, 573–576). The author describes the development of a simple apparatus for routine fluidity testing and the application in the foundry of the results obtained. The effects of composition and temperature on the fluidity of molten iron are briefly discussed. The advantages of the miniature spiral test are enumerated.

Continuous Casting in the Foundry. J. Pascal. (*Met. Constr. Mécan.*, 1953, **85,** Jan., 21–25). The factors limiting the speed of casting in iron foundries are discussed. Advantages accruing from the continuous operation of cupolas are considered.—B. G. B.

Gray Iron Castings. C. O. Burgess. (*Foundry*, 1953, **81,** May, 118–123). The author surveys the production of grey iron castings with special reference to sand, metal, and quality control. The development of shell moulding, the production of nodular iron, and the use of basic-lined cupolas are discussed.—L. E. W.

Production of Diesel-Engine Castings in Grey Iron. J. R. Charlton. (*Inst. Brit. Foundrymen*, June 16–19, 1953, *Advance Copy No.* 1064). A detailed description is given of the methods used at the Wear Winch and Foundry Co. Ltd. for the production of diesel engine frame and cylinder block castings in grey iron.—B. C. W.

Compressed Air Risers. A. Janaszewski. (*Przeglad Odlewnictwa*, 1953, **3,** 4, 120–123). [In Polish]. The application of compressed air to risers is described.—V. G.

The Physical and Mathematical Basis for Calculations of the Cooling of Metal Rods Made by the Continuous Casting Process, with the Help of Differential Equations. H. Klein. (*Giesserei Technisch-Wissenschaftliche Beihefte*, 1952, Mar., 441–454). The physical characteristics used in mould and continuous casting are compared, and the mathematical methods available for the calculation of the cooling processes reviewed. The derivation of the differential equations for heat conduction is described and the characteristics of the thermal properties of casting metals discussed. The boundary conditions are described and from them the heat flow across the surface of the rod calculated.—R. J. W.

Compressed Air Lines and Compressed Air Consumption in a Foundry Wishing to Change Over. R. Brinkmann, W. Wolfer, and W. Gesell. (*Giesserei*, 1953, **40,** Apr. 2, 171–174). Numerical examples are given of the compressed air requirements of sand blasting machines and ramming machines. The effect of the length and diameter of the air line is calculated and allowance made for leakage and pressure loss. Attention is given to the nozzle diameters, to the efficiency of the compressor, and to the amount of cooling water used.—R. J. W.

Crawley Foundry of A.P.V.-Paramount Limited. (*Engineering*, 1953, **175,** May 29, 673–674). Stainless steel castings are part of the production of the foundry of the A.P.V. Company, Ltd., which is described. The stainless steel foundry is equipped with an electric-arc furnace and three high-frequency furnaces. An important feature of the foundry is the sand-treatment and distribution plant.—J. MCK.

The Ajax Pump Foundry. J. Kellett. (*Inst. Australian Found.*, Victorian Div., 1952 Convention, 185–195). The author describes the results of partial mechanization of a small grey iron foundry producing pump parts. Present monthly production is 45 tons of castings ranging from 4 oz. to 1½ cwt. each.—P. M. C.

Ford's Cleveland Foundry. (*Foundry*, 1953, **81,** Mar., 138–187). A detailed and fully illustrated description is given of the new Ford Motor Company's foundry at Cleveland, Ohio. The present daily melting capacity is 1800 tons from ten hot-blast cupolas. Core making, moulding, melting, cleaning castings, and ventilation are dealt with in separate sections.

Segregation Due to Pressure during the Freezing of Grey Iron. G. Mayer. (*Met. u. Giesserei Techn.*, 1952, **2,** July, 230–232). Attempts were made to investigate the freezing characteristics of grey iron by deliberately induced ' pressure segregation ' in internal blowholes. This segregation is described as spheres of liquid metal erupting through a solidified crust. They are caused by the crystallization of graphite from the binary and ternary eutectic mixture. As the phosphorus content increased further ' pressure segregations ' occurred at 965° C.—L. J. L.

Rejects in the Grey Iron Foundry and Eliminating the Causes of Rejects. A. Lincke. (*Met. u. Giesserei Techn.*, 1952, **2,** July, 233–234). The large number of rejects is

attributed to the post-war quality of foundry coke and imported ores. By adding fluorite and silicon to the cupola charge, silicon loss can be prevented, and excess silicon can be obtained which will raise the temperature sufficiently to liberate carbon and enable this to be precipitated later in the form of graphite.—L. J. L.

Faults in Foundry Castings. G. Somigli. (*Fonderia Ital.*, 1953, **2**, May, 451–466 ; June, 499–507). This article, based on work carried out by The International Committee of Foundry Technical Associations, gives an extensive review and classification of foundry faults. These are classified in nine groups ; Irregular growths, cavities, discontinuities, surface defects, incomplete castings, dimensional defects, inclusions, deformations, and invisible faults detectable only by laboratory investigation.—M. D. J. B.

Why Are Defects so Frequent in Cast Compressor Cylinder Blocks. H. Klotz. (*Met. u. Giesserei Techn.*, 1952, **2**, Sept., 329–330). Methods of preventing defects in castings are put forward, taking a compressor cylinder block as an example.

The Use of X-Ray Photography to Detect Flaws Due to Pouring Practice in Casting Steel. W. Förster. (*Met. u. Giesserei Techn.*, 1952, **2**, Aug., 269–272). The author demonstrates with three simple examples that flaws and cavities revealed by X-ray photography are often due to faulty pouring practice, and that these could have been foreseen.—L. J. L.

The Internal Friction (Viscocity) of Molten Metals. M. Nacken. (*Giesserei Technisch-Wissenschaftliche Beihefte*, 1953, Mar., 477–484). The effect of viscocity of metallic fluids on their castability and the way in which the coefficient of viscocity may be determined is first described. The mechanism by which internal friction occurs is illustrated by reference to dislocations in the fluid structure and considerations of the exchange energy. The experimental results on pure metals are presented.—R. J. W.

Inspection in a Mass Production Foundry. S. Pienaar. (Inst. Brit. Foundrymen : *Engineer and Foundryman*, 1953, **17**, Apr., 49). The functions of inspection in a foundry are discussed with special reference to grey and malleable iron founding.—A. D. H.

The Dependence of the Brinell Hardness on the Casting Weight of Cast Irons. K. Wagner. (*Giesserei Technisch-Wissenschaftliche Beihefte*, 1952, Mar., 489–491). The Brinell hardness depends not only on the chemical composition, but is a function of the wall thickness and the casting weight. Differences of up to 40 Brinell hardness can be effected by altering the gross casting weight, the smaller the casting the higher the Brinell number. Experiments are described wherein the hardness is altered by varying the degree of saturation, keeping other variables such as the wall thickness constant.—R. J. W.

Foundry Casting. J. R. Tuckerman. (*Australian Foundry Trade J.*, 1953, **4**, Jan., 5, 7, 9, 11–13). The necessity for dividing costs between the many separate processes in foundry work is emphasized and discussed and methods of costing each operation are described.—P. M. C.

Cost Ascertainment, Analysis and Reduction—With Examples from Foundries. B. C. Carlberg. (*Gjuteriet*, 1953, **42**, Feb., 29–37). [In Swedish]. On the basis that cost reduction can be ensured only by co-operation between production and costs departments, the author shows how economies may be effected and productivity increased.—G. G. K.

Staveley Foundry Training Centre. (*Foundry Trade J.*, 1953, **94**, June 18, 707, 714). The training centre for moulders and coremakers organized by the Staveley Iron and Chemical Co., Ltd., Chesterfield, is described.—R. A. R.

HEATING FURNACES AND SOAKING PITS

The Significance of Fuel Economy for Hot Working Operations in a Works Making Special Steels. W. Riemann. (*Stahl u. Eisen*, 1953, **73**, May 21, 721–727). The problems associated with the fuel economy of heating operations before hot working processes on a variety of steel types are discussed. Experiments have been carried out on heating times and recommendations as to furnace construction are given.—J. P.

The Process of Heating a Metal Charge in Industrial Furnaces. (*Hutnik* (Katowice), 1953, **20**, 3, 93–103) [In Polish]. The mechanism of the heating and soaking of a charge in relation to the type of furnace is outlined. The heating of small and large charges in a constant temperature medium is discussed. The time necessary to heat any charge to a given temperature is calculated. Two nomograms for the calculation of heating time for ingots up to 300 mm. in dia. are given.—V. G.

HEAT-TREATMENT AND HEAT-TREATMENT FURNACES

Heat-Treating Round-Up for 1952. H. E. Boyer. (*Steel Processing*, 1953, **39**, Jan., 30–36). The author describes several new furnaces and their applications to various types of heat-treatment. A new controlled atmosphere furnace incorporating a quench tank, and others for nitriding, and carbonitriding, are described and illustrated. The merits of salt-bath heating and " liquid flame " hardening are discussed.

International Metal Exhibition at Detroit. J. G. Hofman. (*Metalen*, 1953, **8**, Jan. 31, 26–30 ; Feb. 15, 51–54 ; Feb. 28, 70–74). [In Dutch]. The author mentions some of the heat-treatment developments using controlled atmospheres. Special mention is made of the Lindberg Hyco gas producer for full annealing of high carbon steels. Details are also given of the Hyen producer for annealing carbon steels with its modification using CH_4 and CH_4 plus NH_3 for carbonitriding, the Hyex producer for decarburizing, and the Hyam producer for annealing stainless steel. Other heat-treating furnaces and degreasing plant, quantometers, and vacuum fusion apparatus which were exhibited are described.—R. S.

Improvement of Quality of Rolls and Machine Components by Heat-Treatment. J. Cincarek. (*Met. v. Giesserei Techn.*, 1952, **2**, Dec., 446–448). The author gives practical examples for the two-stage normalizing of cogging rolls, the surface hardening of pulley wheels, and the water hardening of driving shafts.—L. J. L.

Gas Carburizing of Substitute Steels. P. Parfitt. (*Machinist*, 1953, **97**, May 9, 745–748, 752). Results obtained in the gas carburizing of two low-alloy Ni–Cr–Mo steels (En 351 and 355) are presented for non-standard specimens $\frac{1}{2}$ in. in dia. It is shown that it is dangerous to rely exclusively on properties derived from the standard $1\frac{1}{8}$-in. dia. specimen.—R. A. R.

Gas Carburizing. (*Automobile Eng.*, 1953, **43**, Apr., 149–151). Methods employed by the Rover Co., Ltd. for hardening crown wheels and pinions are described and illustrated. Prepared town's gas carburizing equipment, manufactured by Wild-Barfield Electric Furnaces Ltd. is used, which incorporates a Holmes-Maxted catalytic sulphur removal plant, and a preparation unit for removing the deleterious carbon dioxide, oxygen, and heavy hydrocarbons. The carburizing furnace takes a retort 24 in. in dia. by 48 in. deep, which holds 75 crown wheels or 180 bevel pinions per charge. Active carburization extends over $2\frac{1}{2}$ hr. The process is more economical than pack carburizing.—P. M. C.

Some Aspects of Gas Carburizing and Carbo-Nitriding. C. H. Luiten. (*Metalen*, 1953, **8**, Mar. 14, 91–97). [In Dutch]. The effects of CO, CH_4 and NH_3 in gas carburizing and carbonitriding are discussed. Carburizing conditions depend on the rate of reactions between the gases, and this also controls soot formation.—R. S.

Heat-Treatment of Newer En Steels. R. Wilcock. (*Metal Treatment and Drop Forg.*, 1953, **20**, Mar., 113–118). The heat-treatment of the newer En steels, which have been developed to conserve alloying elements, is surveyed. Compositions, treatments, and mechanical properties are dealt with for direct hardening, case-hardening, and carburizing types. Views of other metallurgists on this problem are appended.—P. M. C.

The Salt Bath—A Production Tool. L. F. Spencer. (*Steel Processing*, 1953, **39**, Mar., 120–130, 144). The advantages of salt-bath heating are discussed, and salt compositions for different working temperatures are tabulated. Various types of salt-bath heating equipment are described, *viz.*, externally heated, immersion heating element, and immersed electrode. Several mechanisms, both static and conveyor types, for handling the workpieces are described, and techniques for liquid cyaniding, carburizing, nitriding, neutral hardening, and isothermal transformation are outlined.—P. M. C.

Salt Bath Carburizing. J. K. McIver. (*Metal Progress*, 1952, **62**, Dec., 85–87). The use of salt-bath carburizing for case hardening, by a firm manufacturing a variety of products is described. Control of case depth and case composition is discussed ; specifications are met for case depths from 0·0005 to 0·060 in.—B. G. B.

FORGING, STAMPING, DRAWING, AND PRESSING

Looking Ahead with the Forging Industry. J. C. McComb. (*Steel Processing*, 1953, **39**, Jan., 17–21, 36, 37). A review is presented of prospects in methods, technology, and plant for the American forging industry in 1953. The revolutionary Chambersburg Impacter, which forges in mid-air between a pair of horizontally opposed rams, is described. Progress in the construction and installation of the massive presses for the U.S. Air Force's heavy press programme is discussed. Methods of reducing die costs by sinking impressions by drop forging, and by the application of hard facing alloys, are considered.—P. M. C.

Steam Losses in Forging Hammers. V. Tatarinov. (*Steel Processing*, 1953, **39**, Feb., 76, 77). The author considers steam losses in stuffing boxes and in operating valves, and gives a formula for the amount of steam escaping through a known clearance. Charts are given which enable the steam loss to be quickly determined if the clearance and working pressures are known.—P. M. C.

Air Force Heavy Press Program Moves from Planning Board into Fabrication. A. S. Cohan. (*J. Met.*, 1953, **5**, May, Section 1, 624–626). Two 50,000-ton and four 35,000-ton presses are being built by Loewy Construction Co. for the U.S. Air Force. The larger presses will have nine main hydraulic cylinders and six press columns, the smaller will have eight hydraulic cylinders and four press columns. The design of the presses and some of the problems in their fabrication are discussed.—G. F.

The Big Squeeze. (*S.A.E. J.*, 1953, **61**, Jan., 29–32). A brief review is given of the heavy press programme being sponsored by the U.S. Air Force to accelerate the production of larger and stronger structural parts for aircraft. Some 400,000 tons of press capacity, comprising 16 units ranging from 8,000 to 50,000 tons for die forging and extrusion are listed.—P. M. C.

Potentialities of Powerful Forging Presses. (*Machinery Lloyd*, 1953, **25**, June, 99–104). The development and application of large presses in Great Britain and the U.S.A. are discussed. A report on a 12,000 ton forging press installed recently at the Redditch works of High Duty Alloys Ltd. is given.—B. G. B.

Large Mechanical Press for the Production of Steel Cartridge Cases. H. G. Hron. (*Machinery*, 1953, **82**, May 1, 799–805). New presses and operations at the plant of Regal Ware Inc., Kewaskum, Wisconsin, for producing 105-mm. steel cartridge cases from 0·30% carbon steel blanks 7⅜ in. in dia. by 0·515 in. thick are described and illustrated. The equipment includes a Bliss 4000-ton press with a six-station indexing table, a 150-ton hydraulic press, a continuous spray pickling machine, and phosphatizing, lubricating, heat-treating, and gauging units.—R. A. R.

Presses and Hammers. W. Engelhardt. (*Technik*, 1953, **8**, Feb., 89–92). The development of mechanical and hydraulic presses, and pneumatic hammers is reviewed with reference to exhibits at the 1952 Leipzig Fair.—J. G. W.

The Production of Forgings by Pressing. A. M. Manssurow. (*Met. u. Giesserei Techn.*, 1952, **2**, Oct., 370–371). This paper, translated from *Automobile and Tractor Industry*, (Moscow), 1951, No. 1, briefly describes the making of cams and valves by die forging.—R. A. R.

Double End Horizontal Press Expedites Shell Manufacture. (*Steel Processing*, 1953, **39**, Feb., 63–65). A brief illustrated description is given of the production of 90-mm. cartridge cases by the Rheem Manufacturing Co., U.S.A. Starting with a disc of 0·30% carbon steel 0·69 in. thick and 8⅞ in. in dia., a series of eight press and auxiliary operations produces a slender shell case 24 in. long. A novel feature of the clearing presses is that a drawing operation is effected at each end of the press stroke, thus eliminating waste time on what would normally be a long unproductive return stroke.—P. M. C.

A Continuous Crankshaft Forging Plant. (*Machinery*, 1953, **82**, May 8, 861–864). The Chrysler Corp. has recently installed a continuous plant for producing crankshafts from billets weighing 106 to 125 lb. The three principal components are the 6000-ton forging press which can operate at up to 35 strokes/min., a rotary billet-heating furnace 32 ft. in dia., and an automatic continuous hardening and tempering installation. The sequence of processes is described in detail.

The Determination of the Forgeability of Steel by Hot Torsion Tests. (*Mét. Constr. Mécan.*, 1953, **85**, Feb., 101–102). An apparatus, developed by the Timken Roller Bearing Co., to carry out torsion tests on heated samples of steels is described. The results obtained are related to their forging properties.—B. G. B.

Costing and Estimating—Application in the Drop-Forging Industry. A. Taylor. (*Metal Treatment and Drop Forging*, 1953, **20**, May, 223–227).

Steel for Forming-Tools. J. Lomas. (*Sci. Eng.*, 1953, **6**, Jan.–Feb., 15–22). Tools for forming are grouped into steels for the plastic deformation of hot metal, for shaping and trimming dies, die-casting dies, punches, chisels, and similar tools employed for work on cold metal. The most suitable steels for each of these are listed.—L. J. L.

Reducer Roll Machine. (*Machinery Lloyd*, 1953, **25**, Feb. 21, 73–74). A new machine for preparing material for forging, which can also be employed in conjunction with hammers, presses or upsetters, is described. After the material has been preformed in this machine it can be finish-forged without reheating.—R. A. R.

Plastic Deformation of Alloyed Cast Irons in Closed Forging Dies. W. Schlegel and E. Piwowarsky. (*Giesserei Technisch-Wissenschaftliche Beihefte*, 1952, Mar., 455–461). Experiments are described and the results tabulated of ten alloyed cast irons. The material and the machines used to carry out the experiments are illustrated in detail. The shape before forging is considered and the previous heat-treatment and heating of the specimens described. The effect of the speed of pressing and of friction is also considered.—R. J. W.

New Ideas Yield Dividends. T. A. Dickinson. (*Steel Processing*, 1953, **39**, Apr., 163–165, 191). Brief details are given of impact stamping methods applied to the production of sheet metal components by Stainless Steel Products Inc., California. By developing and taking advantage of new production methods the company has multipled its annual volume of sales by 50 in eight years.—P. M. C.

Some Applications of Progressive Tools. G. V. Bevan. (Sheet and Strip Metal Users' Tech. Assoc. : *Sheet Metal Ind.*, 1953, **30**, Mar., 181–190). The author discusses principles and the methods of applying them to the choice and design of progressive tools for producing sheet metal components.

A Check List for the Design of Progressive Dies. F. Strasser. (*Steel Processing*, 1953, **39**, Feb., 66–70, 87). The author specifies a long list of checks to be applied in the design of blanking, forming, and progressive dies. The list covers blank and stock, the cutting of die plates and punches, forming members, die set, and auxiliary members.—P. M. C.

Putting the Squeeze on Jet Engine Parts. (*Steel Processing*, 1953, **39**, Mar., 117–119). A brief description is given of various blanking, drawing, forming, and squeezing operations performed by four new Warco presses capable of exerting up to 600 tons pressure. These presses are part of the equipment at the Ryan Aeronautical Co., San Diego, where exhaust cones, and other jet engine components are produced.—P. M. C.

Tin-Box Making—Details of Some Special Tools for Increased Production. G. Taylor. (*Sheet Metal Ind.*, 1953, **30**, Apr., 277–280). The author gives details and drawings of four special tools : (1) A combination die for the production of round and rectangular solid-drawn containers, which also forms the flange in preparation for double seaming the tops ; (2) an open trimming tool with positioning pad in the top tool and positioning and correcting rim in the bottom tool ; (3) a tool for shaping inverted levers at high speed ; and (4) a double seaming tool containing a lifter plate.—P. M. C.

Cold Extrusion. (*Automobile Eng.*, 1953, **43**, May, 187–189). A résumé is given of recent American developments in the cold extrusion of steel parts. Mechanical and hydraulic presses up to 4000 tons capacity are being built and used to produce artillery shells, ball bearing races, hollow shafts, small gears, and heavier ordnance shells in sizes up to 155 mm. and 76 lb. in weight. Both plain carbon and chromium steels are being extruded on a production basis, and the desirability of initial hardness below Rockwell B60 is stressed. Pressures up to 400,000 lb./sq. in. may be necessary.—P. M. C.

Stainless Steel Tubes Extruded at U.S. Steel's Gary Plant. (*Indust. Heating*, 1953, **20**, Apr., 695–700). A description of a new plant for the production of seamless stainless steel tubes at Gary, U.S.A. by the Ugine Sejournet hot extrusion process is given.—B. G. B.

Cold Extrusion Offers Production Economics. H. Schultz. (*Machine Shop Mag.*, 1953, **14**, Apr., 150–155). The history, and economic, metallurgical, and mechanical aspects of cold extrusion are reviewed. Forward and backward extrusion processes are explained. The properties of 0·10%, 0·20%, and 0·40% carbon steels before and after various degress of reduction by cold extrusion are compared. Some English and American phosphate coatings and lubricants are listed. The Heintz Manufacturing Corp. has found that zinc phosphate is not only the best but also the cheapest coating and the easiest to apply ; the coating should be at least 1 g./sq. ft. of surface. The Mullins " Kold-flo," which needs no special surface lubricant and for which the pressures required are surprisingly low, is mentioned.—R. A. R.

Extrusion Goes Commercial. (*Steel*, 1953, **132**, Apr. 27, 106–108). A brief description is given of the Mullins Kold-flo process. Many parts of complex shape and with good surface, can now be made by cold extrusion with low cost and little wastage of metal.—D. L. C. P.

Processing, Properties, and Application of Cold Finished Carbon Steel Bars. J. F. Byers. (*Iron Steel Eng.*, 1950, **30**, May, 103–107). This paper describes the methods of cold processing employed to obtain the correct combination of properties involved in the production of cold drawn and turned bar stock in carbon steels.—M. D. J. B.

How Draw Speed Affects Stainless Wire. S. Storchheim. (*Machinist*, 1953, **97**, May 23, 844–845). Stainless steel wires 0·016 in. in dia. were drawn through 13 dies down to 0·004 in. in dia., 12 different drawing speeds ranging from 50 to 3900 ft./min. being used. The steels were of the 18/8 type with 0·08% and 0·19% molybdenum and 0·08% of carbon. The heat of friction varied with the speed and caused different amounts of austenite to transform to martensite. Curves show the changes in coercive force, remanence, tensile strength, and electrical resistivity with the drawing speed.—R. A. R.

A Practical Review of Carbide Drawing Dies. L. E. Sanderson. (*Wire Prod.*, 1953, **2**, May, 19–21). The design, use, and maintenance of sintered carbide dies for wire drawing are discussed.—J. G. W.

The Interferometric Gauging of Fine Wire. C. P. Fox. (*Wire Prod.*, 1952, **2**, Dec., 6–7). The principle of interferometry applied to gauging fine wire is explained.—J. G. W.

Some Investigations into Wire Drawing. F. C. Thompson. (*Berg- hüttenmänn. Monatsh.*, 1953, **98**, Feb., 21–31). A survey of the principles, practice, and problems of wire-drawing is given with examples from non-ferrous practice.—R. A. R.

Testing and Inspecting the Working Faces of Hollow Tools, particularly Reducing Dies for the Production of Screws and of Carbide Drawing Dies. W. Lueg. (*Stahl u. Eisen*, 1953, **73**, May 7, 621–629). An apparatus is described for viewing the inside of hollow deforming tools. A conical reflecting surface is inserted axially into the tool and the reflected image viewed with an optical microscope. Photographs show the appearance of new, worn, and defective dies.—J. P.

Modern Lighting for Workshops in the Wire Industry. F. Baierl. (*Draht*, 1953, **4**, Feb., 61–73).

ROLLING-MILL PRACTICE

Calculating Rolling Temperatures. S. Matthes. (*Met. u. Giesserei Techn.*, 1952, **2**, Oct., 356). The author develops formulæ for calculating the loss of temperature during rolling processes.—R. A. R.

Problems in the Construction of Electrical Drives for Reversing Mill Trains. W. Mönnig. (*Met. u. Giesserei Techn.*, 1952, **2**, July, 235–241). Production can be increased by cutting down reversing time. This is achieved by reducing armature diameters, using twin motors and twin drives, and employing amplidyne generators to supply roller drives and control generators.—L. J. L.

A Theoretical Study of the Ward-Leonard and Ilgner Systems. R. Doudou. (*Techn. Human*, 1952–53, **5**, 222–258). The theory of the Ward-Leonard and Ilgner systems of control is presented. The advantages and disadvantages of each system are listed and common practical applications described.

A Modern Drive for Plate and Cogging Mills. P. Clemens. (*Met. u. Giesserei Techn.*, 1952, **2**, Dec., 445). The advantages of a twin drive as compared with normal drives for cogging mill stands are enumerated.—L. J. L.

Comparison of Methods of Calculating Roll Pressures During Cold Rolling. K. Filasiewicz, Z. Wusatowski, and A. Galanty. (*Prace Instytutow Ministerstwa Hutnictwa*, 1953, **5**, 2, 57–81). [In Polish]. A critical survey of formulæ and of methods of measuring roll pressures has been made by comparing calculated values with measured pressures. Roll pressure data are taken from the literature. The best results are obtained when calculating the roll pressure by Siebel's method and the S.K.F., method. In none of the known calculating methods is the effect of rolling speed on roll pressure taken into consideration.—V. G.

Electrically-Driven 42½-in. Cogging Mill. (*Engineering*, 1953, **175**, May 15, 633–635). A cogging mill at the Redbourn works of Richard Thomas and Baldwins, Ltd. which has rolls with 42½-in. centres and a 9-ft. barrel, and its electrical equipment are described. Power for the main mill motor is obtained from a Ward-Leonard Ilgner motor-generator set. The 5000-h.p. main motor is of the shunt-wound compensated type. The control system is described in detail.—J. McK.

Comparison of Conditions of Utilization of European and American Blooming Mills. M. F. Hauth. (*Centre Doc. Sider. Circ. Inform. Tech.*, 1952, No. 9, 1283–1291). The main differences in practice are (*a*) the more frequent supply of smaller ingots in European mills served by Bessemer steelworks, and (*b*) the nature of the finishing mill which in American is generally a continuous strip mill. The influence of secondary local factors is also considered. Rapid Bessemer casting economizes in consumption of fuel for soaking pits, but the composition of the ingots varies.—A. G.

Development of Flat-Rolled Products by Continuous Rolling. S. E. Graeff. (*Iron Coal Trades Rev.*, 1953, **166**, Apr. 3, 759). Developments in the production of flat-rolled steel products by continuous rolling are briefly reviewed, and the increase in production is indicated.—G. F.

Pass Design for Hexagons. H. E. Muller. (*Iron Steel Eng.*, 1953, **30**, May, 55–63). The author describes roll pass design for the production of hexagon bars and examines passes parted at the flats, and finishing pass parted at the corners. Roll pass schedules and layout diagrams are given.—M. D. J. B.

Improvements in Plate Quality to Promote Our Plate Production. W. Dönau. (*Met. u. Giesserei Techn.*, 1952, **2**, Oct., 357–359). The author deals briefly with defects encountered when rolling plates and suggests means of eliminating them.

Calculation of Roll Force in Cold Strip Rolling. P. O. Strandell and A. Leufvén. (*Jernkontorets Ann.*, 1953, **137**, 3, 100–114). [In Swedish]. Experiments which sought to confirm Palmgren's work on roll force when rolling three carbon steels (0·10, 0·60 and 1% C) were extended to include an 18/8 stainless steel. Simplified calculations made use of roll force coefficients (force per unit strip width and roll diameter). Diagrams show these as functions of the ingoing strip thickness divided by roll diameter, and curves are given for pass reductions. It was impossible to establish a general diagram for all qualities based on the yield point. Better results may be achieved by using the ' compression resistance ' as defined by B.I.S.R.A. Future tests will evaluate the effects of speed, tension, and torque on roll force.—G. G. K.

Innovations in the Field of Cold Rolling of Strip Abroad and the Present Position of Processing Strip into Special Sections. K. Wallmann. (*Stahl u. Eisen*, 1953, **73**, May 7, 639–646). The problems associated with the rolling of steel strip received in the pickled and oiled state, such as storage and corrosison prevention, are discussed. The increasing use of the Sendzimir mill is referred to and the design of mills for rolling special profiles are illustrated.—J. P.

The Process of Cold Rolling Steel Strip. J. Martinez Gonzalez. (*Anal. Mec. Elect.*, 1952, **29**, July–Aug., 185–195). [In Spanish]. The author describes scale-breaking machines, chemical descaling methods, and the regeneration of pickling bath solutions. A general description is given of annealing processes and plant.—R. S.

The Variation of Speed during the Cold Rolling of Strip. (*Mét. Constr. Mécan.* 1953, **85**, Feb., 120–123). A number of devices for synchronizing the speed of the two reels on each side of the rollers during the cold reduction of strip are discussed. The Colombes-Textrope, which has recently been developed to maintain synchronization, is described ; the results were very satisfactory.—B. G. B.

Hot Rolling of Wide Strip. F. Wiesner. (*Hutnické Listy*, 1953, **8**, 3, 134–138). [In Czech]. An analysis is made of the advantages and disadvantages of the continuous and interrupted methods of rolling sheet steel under Czechoslovakian conditions.—P. F.

MACHINERY FOR IRON AND STEEL PLANT

Mechanical Conveyors and Their Applications in Industry. N. Andreassen. (*Tekn. Ukeblad*, 1953, **100**, Apr. 23, 351–358). [In Norwegian]. An overall survey of material-conveying equipment describes and illustrates rubber, steel link and mesh type conveyor belts ; roller, chain and overhead conveyors ; and chutes and skip hoists.—G. G. K.

Mechanized Hot Scarfing. J. H. Zimmerman. (*Iron Steel Eng.*, 1953, **30**, May, 108–112). The advantages of mechanized hot scarfing are briefly described, after which the author enumerates the products which can be scarfed. A modern machine comprises a frame with two rows of oxy-acetylene burners which ' scarfs ' or ' deseams ' simultaneously the top and bottom surfaces of slabs up to 72 in. wide.—R. A. R.

The Magnetic Amplifier and Its Application in the Steel Industry. W. E. Miller. (*Iron Steel Eng.*, 1953, **30**, May 65–75). The author reviews the principles of operation and application of magnetic amplifiers as regulators and considers the future possibilities of this equipment. Magnetic amplifier characteristics, basic circuits and current measuring reactors are discussed. These magnetic amplifiers will not, in the immediate future, replace rotating amplifiers or regulators, but should be considered as an additional method with advantages on some applications.—M. D. J. B.

Over-Crank Guillotine Shear for Precision Work. (*Engineering*, 1953, **175**, May 29, 697). The machine, made by Joshua Bigwood and Son, Ltd., Wolverhampton, is capable of shearing mild-steel plate up to $\frac{5}{8}$ in. thick and 4 ft. wide. It is fitted with hydraulic hold-down gear, and adjustable back and front gauges are provided. The maximum speed is 25 strokes/min.—J. McK.

Plate Shears and Their Maintenance. (*Mét. Constr. Mécan.*, 1953, **85**, May, 391). The maintenance of plate shears in France is briefly considered.—B. G. B.

LUBRICANTS AND LUBRICATION

Colloidal Graphite as an Aid in Metallurgy. A. Humann. (*Met. u. Giesserei Techn.*, 1952, **2**, Oct., 351–352). The author extols the virtues of graphite in colloidal suspension. If 0·05 to 1% of colloidal graphite is added to a normal lubricating oil an adsorption of colloidal graphite on the metal surface will take place, thereby producing a smoother gliding surface.

Tests for Ball and Roller Bearing Greases. H. A. McConville. (*Product Eng.*, 1953, **24**, Mar., 160–163). This is a brief survey of testing procedures that are significant in estimating the ability of ball and roller bearing greases to perform satisfactorily in particular services. Tests include those for softening, chemical stability, high and low temperature applications, lubrication, friction, oxidation resistance, and oil liberation.—A. M. F.

On the Solution of the Reynolds Equation for Slider-Bearing Lubrication. II—The Viscosity a function of the Pressure. A. Charnes and E. Saibel. (*Trans. Amer. Soc. Mech. Eng.*, 1953, **75**, Feb., 269–272). The solution of the Reynolds equation for slider bearing lubrication is given for the finite bearing, side leakage included, where viscosity is a known function of pressure. The solution for pressure distribution can be obtained from the corresponding case of constant viscosity.

WELDING AND FLAME-CUTTING

Physical Metallurgy in Modern Welding. L. Pottecher. (*Cien. tecn. Solda*, 1953, **3**, Jan.–Feb.). [In Spanish]. The author broadly surveys the methods of physical metallurgy, including X-rays, gamma rays, in their application to the study of welding processes and weld metal.—R. S.

Low Hydrogen Welding Rods. J. D. Fast. (*Philips Tech. Rev.*, 1952, **14**, Sept.–Oct., 96–101). Electric welding rod coatings, which transmit very little oxygen to the metal during the welding process, suffer from the disadvantage that large quantities of equally undesirable hydrogen are absorbed by the metal from the water vapour present in the coating. In order to decrease this hydrogen pick up, basic or low-hydrogen electrodes containing very little water vapour have been developed. (11 references).—P. M. C.

Ferritic-Austenitic Welding Electrodes of High Heat Resistance. J. P. Wienberger. (*Radex Rundschau*, 1952, No. 2, Apr., 49–75). Details and data are given of extensive series of tests carried out with welding electrodes of various compositions to ascertain whether nickel could be replaced by manganese. The effects of nitrogen were also studied. For best qualities, nickel was found indispensable but 0·25% nitrogen permitted some nickel saving. Manganese adversely affected the scaling resistance. For ferritic-austenitic welding electrodes of high scaling resistance the following composition is suggested : 0·1% C, 0·8–1·0% Si, 3% Mn, 22–24% Cr, 1·5% Ni, 0·25% N_2.—E. C.

ESAB Deep-Penetration Electrodes. (*ESAB Revue*, 1952, No. 4, 1–6). Welding data are presented relating to deep-penetration electrodes produced by Elektriska Svetsnings-aktiebolaget (ESAB) together with results of tests on joints made with them.—R. A. R.

Future Problems to be Faced in the Development of Metallic Arc Electrodes. J. H. Paterson. (*Welder*, 1953, **22**, Apr.–June, 29–33). With the standardization of electrode compositions into groups according to their properties, a somewhat new approach to welding problems has to be made. The author outlines the most important variables introduced during the welding process such as, dilution of weld metal with parent metal, presence of trace elements, and rates of solidification and cooling of weld metal.—U. E.

Slag-Metal Interaction in Arc Welding. N. Christensen and J. Chipman. (*Welding Res. Council Bull.*, 1953, Jan., No. 15). A study was made of the interaction between slag and weld metal, under different conditions of arc length, current welding speeds, and types of electrode and base metal. Factors which affect the retention time and average temperature in the weld puddle will affect the distribution of manganese between slag and bath, and welding speed was the only factor of practical importance under normal manual welding operations.—U. E.

The Influence of Lloyd's Register of Shipping on Welding Progress. H. N. Pemberton. (*Welder*, 1953, **22**, Apr.–June, 38–39). A brief survey is given of the activities of Lloyd's Register of Shipping in conducting tests and regulating the application of welding in ships.—U. E.

Development of Weldable High-Tensile Structural Steels. L. Reeve. (*Welder*, 1953, **22**, Apr.–June, 40–44). Post-war developments in weldable structural steels of high yield point and good weldability are described.—U. E.

The " Thermit " Welding Process in Shipbuilding. W. Ahlert. (*Schweissen u. Schneiden*, 1953, **5**, June, 225, 228–233). The Thermit welding process as applied to the welding of ships is discussed. The metallurgical properties and structure of the weld seam were investigated and it is shown that this process can be applied successfully to the welding of heavy sections.—U. E.

The Welding of Equipment in Chromium–Nickel Austenitic Steel. F. H. Keating. (*Welding*, 1953, **22**, Apr.–June, 61–69). The properties of chromium–nickel austenitic welded steel and correct welding procedures for this type of material are discussed.—U. E.

Welding Processes and Procedures Employed in Joining Stainless Steels. H. Thielsch. (*Welding Res. Council Bull.*, 1952, Sept., No. 14). A review is given of published and unpublished data on metal arc welding, carbon arc welding, gas and resistance welding, and the oxygen cutting of stainless steels. (242 references).—U. E.

Hot Cracking of Steel Welds. E. C. Rollason. (*Welder*, 1953, **22**, Apr.–June, 45–53). The causes of cracking, in particular hot-cracking, in ferrous weld metal are described. The influence of welding practice and metallurgical properties and alloying elements are discussed.—U. E.

Resistance Welding—Its Present Achievements and Future Prospects. H. E. Dixon. (Sheet and Strip Metal Users' Tech. Assoc. : *Sheet Metal Ind.*, 1953, **30**, May, 385–392, 396).

The Hardfacing of Rollers by Welding. W. Küntscher and J. Cincarek. (*Met. u. Giesserei Techn.*, 1953, **3**, Apr., 126–129). A description of a hard-facing welding process employing an austenitic Cr–Ni electrode is given. A 3–6 mm. thick skin of wear-resisting steel is welded on to the surface of the roll.—L. J. L.

Hardfacing by Welding. M. Riddihough. (*J. Junior Inst. Eng.*, 1953, **63**, Apr., 213–216). The structure of wear-resisting alloys used for hard-facing is briefly indicated and the four types of structure produced by hardfacing rods are listed.—L. J. L.

New Trends in Tube Making Practice. H. Doern. (*Demag News*, 1952, No. 129, 1–6). In co-operation with an American engineering company, the Demag Company has taken up the manufacture of resistance-weld tube mills of approved design. Recent developments in the field of tube making are reviewed.—B. G. B.

Some Examples of Practical Methods of Eliminating Welding Fumes. (*ESAB Revue*, 1952, No. 4, 9–14). Fans and ducts for removing welding fumes are described and illustrated.

The Contribution of the Universities to Welding Research. D. Hanson. (*Welder*, 1953, **22,** Apr.–June, 34–37). A brief summary is given of some of the outstanding welding problems, and recent and current research work, carried out at the different universities.—U. E.

MACHINING AND MACHINABILITY

Machining—Theory and Practice. K. G. Lewis and W. Milne. (*Machinery*, 1953, **82,** May, 15, 918–924). This article deals with grinding sensitivity, grinding burn, and the machining of steels and cast iron.—E. C. S.

Phase Transformations during Electric-Spark Machining of Metals and Attempts to Establish Criteria of the Observed Interactions. L. S. Palatnik. (*Doklady Akademii Nauk S.S.S.R.*, 1953, **89,** 3, 455–458). [In Russian]. Structural transformations in the surface layers of various metals machined by electric-sparking in air were investigated by X-ray analysis, measurements of microhardness, and spectrum and microanalyses. On the basis of experimental data the mechanism of the phenomena occurring during sparking and their effects are discussed. and a formula is deduced which is proposed as an approximate criterion of the observed interactions.—V. G.

The Use of Metallic Carbides in the Machining of Steel Castings. E. Blanpain. (*Anal. Mec. Elect.*, 1952, **29,** July–Aug., 181–187). [In Spanish]. The author uses the example of the machining of a wagon wheel to illustrate the advantages of carbide tools over high-speed steel for machining which necessitates roughing and finishing operations.—R. S.

An Electrical Roughness Tester for the Workshop. G. W. van Santen. (*Philips Tech. Rev.*, 1952, **14,** Sept.–Oct., 80–86). A portable roughness tester is described comprising pick-up, amplifier, rectifier, and moving coil meter. The pick-up is moved across the surface by hand, and a sapphire stylus thereby affects a piezo-electric element of barium titanate. The alternating voltage obtained, is amplified, and rectified, and the mean value is measured by the moving coil meter.—P. M. C.

POWDER METALLURGY

Micropowder Magnets. (*Engineering*, 1953, **175,** May 29, 682–684). **Iron Powder Magnets.** (*Engineer*, 1953, **195,** May 22, 740–741). The manufacture and properties of permanent magnets made from soft ferromagnetic powders are described. These magnets are moulded under pressure, the raw material being very fine powders of iron or iron alloys, known as micropowders. The micropowders are produced by low-temperature chemical reduction of iron compounds. The fineness of particle size has an important effect on the magnetic properties obtained.—J. MCK.

Chromium–Nickel Alloy Steel Sinterings: An Investigation of Physical Properties. J. D. Shaw, W. V. Knopp, and B. A. Gruber. (*Precision Metal Molding*, 1953, **11,** Mar., 42–45, 73–76). Results are reported of an investigation of the properties of powder metallurgy products made by sintering at 1100° C. for 1 hr. in hydrogen chromium-nickel alloy powder mixed with 5%, 10%, and 20% of electrolytic iron powder. Three grades of chromium-nickel alloy, namely 80/20, 50/50, and 20/80, were used, and in some cases the iron contained 1% of graphite.—R. A. R.

The Recovery of the Components of Sintered Carbide Scrap. B. Zacharzewski and C. Lésniak. (*Hutnik* (Katowice), 1953, **20,** 3, 83–93). [In Polish]. A dry method of recovering sintered carbides from scrap is described. It is claimed that not only can tungsten carbide. titanium carbide and cobalt be recovered, but also carbides of thalium, vanadium, niobium, and molybdenum. The method entails oxidation of the scrap at 800–900° C. in air, crushing and screening of oxidised material, reduction in hydrogen at 700–1000° C., carburization of the reduced material by mixing with soot and heating in hydrogen at 1450–2000° C. (depending on chemical composition). The chemical basis of the process is outlined. Sintered carbides produced from regenerated powders had properties equal to those from fresh powder, providing at least 2% fresh cobalt was added and the mixture ground in water for the usual period.—V. G.

The Manufacture of Metal Powders by Mechanical Means. H. Kramer. (*Metall*, 1953, **7,** Apr., 262–268). Various mechanical methods of pulverising ductile metals are examined, and a cyclone mill, its chacteristics and basic principles are discussed.—J. G. W.

Bimetallic Sintered Metal Parts. V. V. Saklinskii. (*Vestnik Mashinostroeniya*, 1952, **32,** 3, 76–79). Method of production of bearing linings and bushes from metallic powders are described in some detail. Savings in non-ferrous metals up to 80%, and in labour up to 66%, are claimed.—V. G.

PROPERTIES AND TESTS

The Behaviour of Low Carbon Basic-Bessemer and Open-Hearth Rimming Steels during Tensile and Notched-Bar Impact Tests. O. Hengstenberg and W. Jäniche. (*Stahl u. Eisen*, 1953, **73,** June 18, 828–838). Tests on a number of basic-Bessemer and open-hearth rimming steels, as rolled, naturally and artificially aged, and with different degrees of deformation and heat-treatment, revealed the effects of phosphorus and nitrogen on the tensile and impact strengths. Nevertheless, for the steels with the same carbon, phosphorus and nitrogen contents, variations occur with cannot be correlated with oxygen content or secondary grain size. It is concluded that, besides phosphorus and nitrogen, there is some unknown factor which influences these mechanical properties.—J. P.

Statistical Analysis of the Mechanical Properties of Standard Structural Carbon Steels. W. Mosznyski. (*Hutnik* (Katowice), 1953, **20,** 2, 41–49). [In Polish]. The use of statistical analysis for the investigation of the variation in properties of standard structural carbon steel is discussed.—V. G.

Mechanical Testing Equipment for Sheet and Strip Metal. G. Földes. (*Sheet Metal Ind.*, 1953, **30,** May, 357–364). Descriptions and illustrations are given of many of the instruments and equipment manufactured by Alfred Amsler and Co., Schaffhausen. The article covers tensile, hardness, cupping, ductility, and fatigue tests.—P. M. C.

Amsler Repeated Impact Testing Machine. D. Iturrioz. (*Met. Elect.*, 1953, **12,** Mar., 37–40). [In Spanish]. The Amsler repeated impact testing machine and the various types of test piece are described.—R. S.

Impact Ductility of Soft Steels with Varying Aluminium Contents. Å. Josefsson and E. Nygren. (*Jernkontorets Ann.*, 1953, **137,** 3, 69–99). [In Swedish]. The effects of increased aluminium content on brittle fracture and grain size were studied in three series of steels having 0·15% and 0·18% C. Charpy tests were carried out at various temperatures with and without straining and ageing. The point of transition to brittle fracture was used as a measure of ductility. Additions of aluminium up to 0·3% refined the grain size and reduced the transition temperature.—G. G. K.

The Problem of Brittle Fracture. C. F. Tipper. (*Welder*, 1953, **22,** Apr.–June, 54–61). The brittle fracture occurring in structural mild steel and the effect on weldability are discussed. Although service fractures have often started in defective welds they do not usually follow the welds nor do they follow those zones which have been plastically deformed during the contraction of the weld metal on cooling. Any stresses which can be attributed directly to welding are unlikely to be so severe as those due to bending, flanging, or shearing.—U. E.

The Mechanics of Notch Brittle Fracture. A. A. Wells. (*Welding Res.*, 1953, **7,** Apr., 34*r*–56*r*). A method is described which makes it possible to measure the energy absorbed in plastic flow along the surfaces of a propagating brittle fracture by means of the temperature wave created. A notch-brittle material has been shown experimentally to possess, at a given temperature, a minimum surface energy of crack propagation. In general, it appears that the brittle breaking loads of large specimens are determined by the initiation process, where the influence of stresses other than those due to external loads is excluded, while the breaking loads of small specimens are controlled by minimum surface energy.—U. E.

Residual Stresses Introduced During Metal Fabrication. K. R. Van Horn. (*Trans. Amer. Inst. Min. Met. Eng.*, 1953, **197**; *J. Met.*, 1953, **5**, Mar., 405–422). The author reviews the field of residual stresses in metals and discusses their beneficial and injurious effects. Examples from commercial practice are given to illustrate the location of these stresses and the practical effects which they introduce. Methods of measuring residual stress are assessed and the relief of such stresses by thermal and/or mechanical treatment is discussed.—G. F.

Stress Distribution and Strength Condition of Two Rolling Cylinders Pressed Together. E. I. Radzimovsky. (*University of Illinois Engineering Experiment Station, Bulletin Series No.* 408, 1953, **50**, Feb.). The results of an investigation into the stress conditions in the contact zone of two compressed cylinders when rolled together with their axes parallel, are given. The manner in which the stress components change during the loading cycle at the various layers in the contact zone were investigated.—B. G. B.

A New Method for Determining the Heat of Activation for Relaxation Processes. C. Wert and J. Marx. (*Acta Metallurgica*, 1953, **1**, Mar., 113–115). The utility of the linear relationship between the heat of activation of a relaxation process and the temperature at which the maximum relaxation occurs is pointed out.—A. D. H.

The Plate Analogy as a Means of Stress Analysis. J. J. Ryan. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 7–28). The author explains the theory of the solution of complex stress problems by using the plate analogy, and describes experiments carried out to determine the applicability of the analogy as a means of stress analysis. The results were unfavourable.—P. M. C.

Improved Photogrid Techniques for Determination of Strain over Short Gauge Lengths. J. A. Miller. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 29–34). Methods are described for photographing on metal a grid of well defined lines 0·0006 in. wide space 0·01 in. apart. To obtain this precision a master grid was ruled for the purpose. Excellent definition of lines was obtained by using cold top enamel and "Dyrite" black contact emulsion. Applications of the method to the study of strain concentrations in the plastic range are described.—P. M. C.

Some Phases of the Technique of Recording Performance Data on Large Machines. J. H. Meier. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 35–52). The advantage of reference signals in bridge work is stressed, and formulæ are given for relating reference signals to the corresponding strains. A shear force measuring device and torque wrench are discussed as examples of the versatility of Wheatstone bridge arrangements. A straightforward method is described and illustrated for calculating strain when the various gauges in a bridge circuit have different orientations with respect to the stress field. Scanning devices for individual gauges are discussed, bridge control circuits are reviewed, and a direct recorder is described.—P. M. C.

Stress Concentrations Produced by Multiple Semi-Circular Notches in Infinite Plates under Uniaxial State of Stress. A. J. Durelli, R. L. Lake, and E. Phillips. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 53–64). A photoelastic solution is given for the state of stress caused by from one to five semi-circular notches for the case of uniaxial tension parallel to a plate edge.—P. M. C.

The TMB Strain Cycle Gauge and Counter. N. H. Jasper. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 87–96). The author describes an instrument for the statistical determination of the strain history of structures, developed by the Taylor Model Basin, U.S.A. The installation, which comprises a mechanical gauge and an electrical counter, will automatically analyse the strain history at the location of the gauge in terms of the number of cycles of given amplitudes that have occurred.—P. M. C.

Residual Stresses in Surface-Hardened Oil Field Pump Rods. R. E. Hanslip. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 97–112). An account of the method of determining the residual stresses in a solid round rod by removing concentric layers of metal from the outside and boring the inside, based on the method of George Sachs, is given.—P. M. C.

Determination of Dynamic Stress-Strain Curves from Strain Waves in Long Bars. W. R. Campbell. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 113–124). A method is presented for experimentally deriving the relation between stresses and strains set up in a long bar by a deformation being propagated as a longitudinal plane wave. Analytical concepts are outlined, and the experimental technique is described in detail.

A Mechanical Deflection Gage—An Instrument for Measuring Displacement Under Impact. D. D. MacLaren, I. J. Taylor, and L. S. Beedle. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 135–146). This gauge has been developed by the Norfolk Naval Shipyard to study the effects of underwater explosions on steel plates. Fundamentally, the gauge is a voltage divider consisting of a fibre tube uniformly wound with Nichrome wire across which a fixed voltage is applied. The tube is fastened to the test plate by a swivel bracket and is supported at the reference bulkhead by a flexible assembly, containing the contact of the voltage divider, through which the tube can pass during movement of the test plate. During dsplacement of the test plate the potential between the reference bulkhead contact and one end of the fixed resistance will vary as the wound fibre tube moves along the contact. This voltage is amplified and recorded. The construction of the gauge, its principles and operation, and brief analyses of typical records are discussed.—P. M. C.

Measurement of the Resistance of Materials to Mechanical Shock. T. E. Pardue and B. Goldberg. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 147–156). Experiments are described in which cantilever test specimens of nodular cast iron and aluminium alloys were subjected to a shock motion consisting of a relatively small number of vibrations of such large amplitudes that considerable plastic deformation was involved. The common mechanical properties are inadequate criteria of shock resistance. The ductility of a material and its ability to absorb energy are among the most significant factors.—P. M. C.

Impulsive Loads on Beams. W. H. Hoppmann. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 157–164). In order to study strains and deflections of a beam subjected to an intensive force by duration short compared with the period of the fundamental mode, a series of hinged beams were investigated both theoretically and experimentally.—P. M. C.

Properties of Bolts Under Shock Loading. H. M. Forkois, R. W. Conrad, and I. Vigness. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 165–178). A programme of experimental work is described which has for its objectives: (*a*) Determination of the loads on, and physical properties of, commonly used types of bolt when subjected to types of shock that may be expected aboard naval ships; and (*b*) determination of good designs and materials for holding-down bolts subject to these shocks. Results are given for $\frac{3}{4}$-in. dia. bolts 2 and $4\frac{1}{2}$ in. lo ng of 0·20% carbon steel.

Elastic and Creep Properties of Stresscoat. W. F. Stokey. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 179–186). Tests were made on two grades of stresscoat; No. 1206 for use at normal temperatures, and No. 1202, which is less brittle, for lower temperatures. Both cast and evaporated specimens were tested as cantilever beams and thin tubes in torsion. Electrical resistance strain gauges were attached to the specimens, and their moduli of rigidity and elasticity measured.—P. M. C.

A Mechanism for Controlling Large-Scale Fatigue Testing Machines. W. J. Hall and G. K. Sinnamon. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 203–208). A 50,000-lb. capacity, lever-type fatigue testing machine has been modified in order that the maximum and minimum loads during the stress cycle, and also the mean load, can be varied according to a predetermined programme. The lever arm ratio is 10 : 1 and the machine operates at 18,000 cycles/hr.—P. M. C.

Longitudinal Impact of Cylindrical Bars. E. A. Ripperger. (*Proc. Soc. Exper. Stress Anal.*, 1952, **10**, No. 1, 209–226). Experiments are described in which the strains developed by the longitudinal impact of bars were measured and recorded by means of strain gauges. (13 references).—P. M. C.

The SR-4 Type Equilateral Fleximeter. G. A. Brewer. (*Proc. Soc. Exper. Stress Anal.*, 1953, **10**, No. 2, 1–8). An instrument is described which permits the determination of mid-plane stresses from external measurements alone, in structures having an inaccessible surface. The Fleximeter, as the instrument is called, determines the state of stress due to flexure alone, and these data, combined with the total outside surface stress as determined by means of a strain gauge rosette, permit the calculation of the mid-plane and finally the inside state of stress on the inaccessible surface.—P. M. C.

Determination of Centrifugal Stresses in a Turbosupercharger Impeller by Means of Stresscoat. W. G. Schmittner. (*Proc.*

Soc. Exper. Stress Anal., 1953, **10**, No. 2, 9–22). Experiments are described in which Stresscoat was successfully used to determine the centrifugal stresses in an impeller rotating at 20,000 r.p.m. The sensitivity of the Stresscoat was determined by means of a four-bladed calibration device for which the centrifugal stress distribution had been calculated.

Application of Optical Interference to the Study of Residual Surface Stresses. H. R. Letner. (*Proc. Soc. Exper. Stress Anal.*, 1953, **10**, No. 2, 23–36). Optical interference offers advantages in the study of residual surface stresses when used in conjunction with the technique of removing uniform layers of stressed material from the surfaces of rectangular bars and measuring the resulting deflections. The mathematical relationship between stress and curvature is developed, and the necessary apparatus and experimental procedure for measuring the latter by optical interference are described. (18 references).—P. M. C.

The Effect of Superposition of Stress Raisers on Members Subjected to Static or Repeated Loads. A. Q. Mowbray, jun. (*Proc. Soc. Exper. Stress Anal.*, 1953, **10**, No. 2, 153–166). Static tensile tests on 0·483 in. thick structural steel plate, and rotary fatigue tests on a Cr–Ni–Mo steel bar were undertaken to assess the decrease in strength produced by compound notches in the form of grooves superimposed on fillets. The experimental elastic stress concentration factor K_e for a grooved fillet under static load was greater than the theoretical values for either the fillet or the groove alone. The stress concentration factor K_e for the grooved fillet ranged from 79% to 89% of the product of the individual factors K_f and K_g for the fillet and groove respectively. There was a tendency for the ratio K_e and K_fK_g to increase with decreasing groove radius.—P. M. C.

The Behaviour of SR-4 Wire Resistance Strain Gauges on Certain Materials in the Presence of Hydrostatic Pressure. W. R. Clough, M. E. Shank, and M. Zaid. (*Proc. Soc. Exper. Stress Anal.*, 1953, **10**, No. 2, 167–176). General effects of hydrostatic pressure and gauge mounting techniques are discussed. Experiments show that when both the dummy and active gauges are under pressure, the mechanical effect on the gauges and on their cement bonds gives rise to an unbalance of the Wheatstone bridge. This unbalance may be linear or non-linear. A pressure calibration method has been used to investigate strains in small grey cast iron and nodular iron pressure vessels. A general correction factor is proposed to compensate for the mechanical effect of pressure.

On the Plastic Behaviour of Metals at Low Temperatures H. Te Gude. (*Metallkundliche Berichte*, 1951, No. 3 : Verlag Technic). The plastic properties of 16 metals, including almost pure iron, were investigated. The terms used, such as yield point, breaking strength, elongation, and hardness, are defined ; experimental apparatus is described ; results are given in tables and graphs.—L. J. L.

Cause and Prevention of Fractures in Bolts. J. Cincarek. (*Met. u. Giesserei Techn.*, 1952, **2**, Oct., 369). Fractures of bolts used in mounting bearings of blooming mill stands can be avoided by using a soft unalloyed steel subjected to a water hardening process rendering its structure more close-grained and improving its ductility.—L. J. L.

A Dual-Amplitude Axial-Load Fatigue Machine. A. E. McPherson. (*Proc. Soc. Exper. Stress Anal.*, 1953, **10**, No. 2, 143–152). A description is given of a fatigue machine which can apply to a specimen a combination of two sinusoidally varying axial loads of different amplitudes (up to 2000 lb.) at frequencies up to 1000 cycles/min. The mean load, which remains constant, can be measured at any time with a maximum error of ± 3%. The variation in measured stress in a given group of specimens is of the order of 1%.—P. M. C.

Fatigue Properties of Springs as Affected by Heat-Treatment. R. L. Rickett and A. O. Mason. (*Metal Progress*, 1953, **63**, Mar., 107–110). The endurance limit of austempered springs was found to be practically the same as for springs of the same tensile strength made from ' oil tempered ' (retempered) wire as for quenched and tempered or martempered springs. At the same tensile strength, however, the limit of proportionality of the austempered springs is lower, hence their tendency to ' set ' under high loads is greater. If the limit of proportionality of the austempered springs is raised to that of the oil-tempered wire, by austempering to a higher hardness and tensile strength, the endurance limit is increased. The same trends were found for specimens of wire tested in reversed bending.—B. G. B.

Instrumentation and Fundamental Experiments in Plasticity. D. C. Drucker and F. D. Stockton. (*Proc. Soc. Exper. Stress Anal.*, 1953, **10**, No. 2, 127–142). A number of basic experiments on the behaviour of thin walled aluminium alloy tubes in the plastic range are described, and the measuring techniques are given in some detail. Of especial interest are : the construction and performance of a mechanical extensometer for measuring axial and circumferential strain ; the behaviour of resistance wire strain gauges in the plastic range ; a successful simple means of making a lever type testing machine indicate load accurately without moving the poise ; and the significance of test results in terms of the mathematical theory of plasticity and experimental stress analysis.—P. M. C.

Hardness and Hardenability. H. E. Boyer. (*Steel Processing*, 1953, **39**, Feb., 78–82). The author discusses in detail the difference between hardness and hardenability, (the capacity to through harden). The roles of carbon in determining hardness, and that of alloying elements in controlling hardenability are described. The Jominy end-quench test and the various structures produced by heat-treatment processes are discussed and illustrated.—P. M. C.

Macrographic Study of Internal Stresses Due to Plastic Deformation. V. Vilhalm. (*Tec. e Indust.*, 1953, **31**, Mar., 245–251). [In Spanish]. The author discusses the origin of internal stresses in the ingot and subsequent stages of steel production, and considers the phenomena of ageing. The production of internal stresses by welding, cutting, impact, bending, and torsion are illustrated, and the effects of these treatments on the mechanical and physical properties are indicated.—R. S.

The Effect of Strain in Artificial Ageing of Steel. F. Van Wijk. (*Metalen*, 1953, **8**, Jan. 15, 1–4). [In Dutch]. The effect of strain in artificial strain ageing is measured by the increase in Charpy impact. Half-inch thick plate was subjected to tensile testing and cold bend tests. The effect on ageing of up to about 3% strain on the impact strength is proportional to the amount of ageing. The results are probably not valid for other thicknesses of material.—R. S.

The Application of the Principle of Raising the Elastic Limit of Steels by Cold Work and Artificial Ageing in the Manufacture of High Pressure Tubes, and the Associated Economic Advantages. G. Ferrand. (*Berg- hüttenmänn. Monatsh.*, 1952, **97**, Oct., 192–201). The use of pre-strain and strain-ageing in addition to the normal processes of auto-frettage, as applied in France recently to high-pressure tubing (made of Confor E steel (C < 0·22%, Mn 0·90%, Cr 0·40%, Cu 0·50%)) for hydro-electric power stations is described. Increased strength and a better safety factor are obtainable than with conventionally prepared tubes.—P. F.

Some Experiments on the Problem of Mechanical Ageing. W. Gilde and E. Benser. (*Met. u. Giesserei Techn.*, 1952, **2**, Dec., 419–422). Attention is drawn to the different ageing characteristics of open-hearth and Bessemer steels ; the impact values for both after annealing are compared. Heat treatment affects ageing only if quenching takes place at temperatures above A_3 ; in samples which were stretched after quenching and then heated to 250° C., notch toughness increased and hardness decreased ; the quenching effect persisted even when the carbon content was reduced to 0·04%.—L. J. L.

Effects of Macrostructure on the Performance of Alnico Permanent Magnets. D. G. Ebeling and A. A. Burr. (*Trans. Amer. Inst. Min. Met. Eng.*, 1953, **197**: *J. Met.*, 1953, **5**, Apr., 537–544). The authors have studied the effects of casting texture, grain orientation and grain size on the performance of Alnico permanent magnets, the technique including the production of single-crystal specimens by special sintering and grain-coarsening processes. In addition to texture and orientation effects, the performance of the permanent magnet is impaired by a refinement of the grain size.—G. F.

Apparatus for the Measurement of the Conductivity of Metals. H. Bock and H. Wagenbreth. (*Technik*, 1953, **8**, Jan., 23–25). An instrument, developed by W. Koch, with which conductivities of different metals may be compared is described. It is shown how the absolute conductivity of reference specimens may be determined by calculation from measurements in a modified version of the same instrument.

Ultrasonic Measuring Instruments. R. H. Warring. (*Machinery Lloyd*, 1953, **25**, June, 69–71). The basic principles of ultrasonic methods of measurement of thickness and the detection of flaws are briefly described.—B. G. B.

The Piezo-Electric Effect and Ultrasonic Waves. F. Solano de Aguirre. (*Dyna*, 1953, **28**, Feb., 39–46). [In Spanish]. The principles of piezo-electricity and the vibration of quartz crystals are briefly described. Piezo-electric circuits are illustrated. Some of the main uses of ultrasonic waves in industry are mentioned.—R. S.

Non-Destructive Testing of Engineering Parts. G. S. J. Read. (*New Zealand Eng.*, 1953, **8**, Jan., 8–16). A survey is presented of the magnetic, supersonic, and X-ray and gamma-ray methods of non-destructive testing. The principle of operation of each is described, and its use and field of application reviewed. There is a brief discussion of the shortcomings of each method and the points necessary for successful application.—P. M. C.

Design and Construction of Betatron Apparatus. I—Principles and History of the Betatron. O. Vaupel. (*Stahl u. Eisen*, 1953, **73**, May 21, 705–706). The principles and development of the betatron electron accelerator and the characteristics of the X-radiation produced are discussed.—J. P.

Design and Construction of Betatron Apparatus. II—Twin-Beam Betatron. R. Wideröe. (*Stahl u. Eisen*, 1953, **73**, May 21, 706–709). The Brown-Boveri twin-beam betatron is described. This has two anti-cathodes and allows the irradiation of two patients at the same time. A development of this apparatus is one in which the two beams travel in the same direction but at a converging angle of 15°. It is thus possible to take stereo-radiographs which prove useful in assessing the depths of defects in solid objects.—J. P.

Design and Construction of Betraton Apparatus. III—A Light Mobile 15-MeV. Betraton. K. Gund. (*Stahl u. Eisen*, 1953, **73**, May 21, 710–713). The Siemens-Reiniger 15-MeV betratron incorporates a novel electron injector and employs a system which enables the weight of the whole apparatus to be reduced. The betatron weighs 880 lb., and is very mobile.—J. P.

Design and Construction of Betatron Apparatus. IV—Irradiation of Thick Steel Samples with the Betraton. M. Komers. (*Stahl u. Eisen*, 1953, **73**, May 21, 713–721). The betatron is particularly suitable for the examination of steel objects greater than 80 mm. in thickness ; some examples of its use are illustrated. In examining assembled components (electric motors, etc.) the hard radiation appears to be better than normal X- or γ-ray methods for detecting faults in parts of heavy section.—J. P.

Gamma Radiography for Non-Destructive Testing. E. G. Fritz. (*Canad. Metals*, 1953, **16**, Apr., 56–58). The author gives a brief history of the use of gamma radiography in the examination of the macrostructiure of metals. Reasons for using iridium^{192} isotope instead of other isotopes are given, and a few applications mentioned.—J. C. B.

An Analytical Theory of the Creep Deformation of Materials. Y.-H. Pao and J. Marin. (*J. App. Mech.*, 1953, **20**, June, 245–252). A mathematical theory of creep is proposed for an ideal material, *i.e.*, one which is homogeneous and isotropic. It takes into account the initial elastic strain, the transient creep strain, and the minimum rate creep strain, and it is capable of representing creep recovery as well as creep. The theory is proposed for simple and combined stress systems and for constant and varying stresses.—J. R. P.

Creep Properties of Steels for Power Plants. A. E. Johnson and N. E. Frost. (National Physical Laboratory : *Engineering*, 1953, **175**, Jan. 2, 25–28 ; Jan. 9, 58–60 ; Feb. 20, 249–250). An investigation was made of the effects of practically possible fluctuations of stress and temperature upon four steels typical of those used in steam-power plant. The steels were : (1) A 0·17% carbon steel ; (2) a 0·5% molybdenum steel ; (3) and (4) a 0·5%-molybdenum 0·23%-vanadium steel in the forms of bar and pipe. The accuracy with which it is possible to forecast the margin of temperature to be allowed in design by analytical means was also examined. The majority of the cyclical fluctuations of stress and temperature dealt with cause increases of creep strain which are within a ratio of 2 : 1 of those given by the basic creep tests. Large fluctuations, of order 50% overstress or 50° F. excess temperature, may modify creep rate and strain to a degree completely outside of these limits. The effect of frequency of these fluctuations is never greater than expressed by a 1·5 : 1 ratio to a mean value ; it is suggested that no serious worry should be caused to the designer by such effects.—R. A. R.

Study of the Mechanical Properties of Steels under Temperature Conditions Similar to Those Obtained During Welding. N. N. Prokhorov and S. A. Kurkin. (*Avtog. Delo*, 1950, No. 10, 6–10). [In Russian]. In the investigation described, the yield strength and plasticity of five steels were determined at temperatures in the 1100–1430° C. range. A special device was used to measure stresses producing hot cracks. The range of compositions of the steels was : C 0·10–1·0%, Cr 0·09–17·0%, Ni 0·09–12·0%, Mn 0·28–1·37%, Si 0·28–1·2%, Ti 0–0·8%, and Mo 0–3·5%. Micro- and macrostructures of specimens are shown.—S. K.

Creep and Rupture of Chromium-Nickel Austenitic Stainless Steels. E. J. Dulis, G. V. Smith, and E. G. Houston. (*Amer. Soc. Met., Preprint*, 1952). Creep tests lasting up to 3000 hr. were made on specimens of 15 Cr–Ni stainless steels, some containing molybdenum, niobium and titanium. Free-machining 18/8 and types 303 and 304 have substantially the same creep strengths at 595°, 705°, and 815° C. Sigma formation and carbide precipitation were the principal microstructural changes observed. Sigma formed most rapidly and most plentifully (up to 15%) in type 316 Nb (Cr 18·50%, Ni 14·19%, No 2·45%, Nb 0·49).—R. A. R.

Development in Production of Alloys. A. E. Williams. (*Min. J. Annual Review*, 1953, May, 105–107). The author discusses a number of recent developments in this field. Among the items considered are nodular cast iron, carbide formation diffusion processes, and the sigma phase in alloy steels, nitriding processes, creep-resisting alloys, and alloys for use at 650–900° C.—B. G. B.

Study of the Problem of Hydrogen in Steel. M. Šícha. (*Hutnické Listy*, 1953, **8**, 1, 14–20 ; 2, 73–80). [In Czech]. Following a detailed survey of previous work on hydrogen in steel, solid and liquid, methods of testing for its presence are described, and, on the basis of the experimental work presented, a new theory on the hydrogen-slag equilibrium is developed. The influence of the nature of the charge, furnace atmosphere, moist materials in the charge, and other factors on the hydrogen content are considered.—P. F.

The Solubility of Oxygen in Gamma Iron. J. A. Kitchener, J. O'M. Bockris, Molly Gleiser, and J. W. Evans. (*Acta Metallurgica*, 1953, **1**, Jan., 93–101). [In English]. The solubility of oxygen at 1325–1423° C. was measured by heating thin strips of high-purity iron foil in a controlled CO–CO_2 gas mixture. Dissolved oxygen was measured by reaction with hydrogen at 1275° C. Three different samples gave maximum solubility values of $0{\cdot}003 \pm 0{\cdot}003\%$ oxygen by weight. The solubility did not vary significantly with temperature. It is suggested that traces of impurities may lead to too high values. It is concluded on this basis that the solubility of oxygen in truly pure γ-iron is $< 0.001\%$. The unfavourable size factor of the O^- is responsible for the low solubility.—A. D. H.

Anelastic Effects in Iron Containing Vanadium and Nitrogen. J. D. Fast and J. L. Meijering. (*Philips Research Reports*, 1953, **8**, Feb., 1–20). The maximum quantity of nitrogen taken up by an iron wire containing 0·5 atm.-% vanadium during heating at 950° C. in nitrogen with 1% hydrogen corresponds to 1 atom nitrogen for 1 atom vanadium plus an extra amount of the same order as the quantity taken up by pure iron under identical conditions. The first amount combines chemically with the vanadium and causes no internal friction. The extra amount causes not only a damping peak corresponding with that in pure iron, but also with a peak at higher temperatures.—R. A. R.

Influence of Alloying Constituents on the Mechanical Properties of Heat-Treated Cast Steel. H. Juretzek, A. Krisch, and W. Trommer. (*Arch. Eisenhüttenwesen*, 1953, **24**, Jan.–Feb., 69–82). Test pieces cast from 283 heats made in a 20-kg. high-frequency furnace have been heat-treated to tensile strengths of 110–130 and 90–110 kg. sq. mm. The steels were singly, doubly, trebly and quadruply alloyed, the additions being chromium, manganese, nickel, molybdenum, vanadium and copper. The test pieces were submitted to tensile and notched-bar impact tests at room temperature and at −50° C. As far as toughness is concerned, there appears to be no advantage in multiple alloying ; the addition of a third or fourth element causing no improvement. The best results were obtained with Cr–Mo steels. Multiple alloying is useful only for such special properties as through-hardening and improved castability.—J. P.

Tellurium in Cast Iron. D. Magri. (*Fonderia Ital.*, 1953, **2**, Apr., 409–411). The author examines briefly the effects of additions of tellurium to cast irons, and studies the depth of temper in relation to total carbon and silica content for additions of different amounts.—M. D. J. B.

Processing of Boron Steels in the Shop. G. D. Rahrer. (*Amer. Soc. Met.* "*Boron Steel*", *2nd revised edition*, 1953, 61–65). The author discusses the machining, cold and hot forming, and welding of boron-treated steels. Boron had no adverse effect on the weldability. The annealing time for boron steel was much less than that for low-alloy Ni–Cr–Mo steel.—R. A. R.

Boron Steel in Through Hardened Parts for Truck Axles. T. A. Frischmann. (*Amer. Soc. Met.* "*Boron Steel*", *2nd revised edition*, 1953, 67–69). Clutch plates, regularly specified as 0·55% Ni, 0·50% Cr, 0·20% Mo steel are now made of 0·30% Ni, 0·40% Cr, 0·12% Mo steel. Axle shafts are now made of 0·55% Ni, 0·65% Cr and 0·12% Mo steel instead of 1·75% Ni, 0·80% Cr and 0·25% Mo, using boron-treated steels. The author comments favourably, on the whole, on these changes.—L. J. L.

Tensile Tests on Flame-Cut Specimens of Cr-V Steel. A. Erker. (*Schweissen u. Sdneiden*, 1953, **5**, May, 183–189). Tensile and fatigue tests were carried out on flame-cut specimens of a Cr–V steel (0·34% C) and the hardness was studied. The effect of heat-treatment and sand-blasting on the properties the flame-cut steel was also investigated. The two parallel cuts did not affect the property of the steel between them when they were more than 0·5 in. apart. The post-heat-treatment of flame-cut steel had a bad effect and was only useful if the sample was heated to dark-red or red heat; it improved the ductility but not the fatigue or tensile strength. Sand-blasting was found to bring the fatigue strength of the heat-affected zone up to that of the unaffected zone of the steel but it did not alter the brittleness. The ductility was in all cases very low.—U. E.

You'll Be Getting Better Electrical Sheet and Strip. N. P. Goss. (*Iron Age*, 1953, **171**, Feb. 5, 147–152, 186–188). The quality of highly orientated polycrystalline silicon steel with the same characteristics of a single crystal is constantly being improved. The desired permeability is closely associated with the composition, non-metallic components, and processing sequence. Maximum permeability commercially available today is about 60,000 but 3·8 million has been reached in 6·4% silicon ferrite by heat-treating in a magnetic field. Application to transformers may produce substantial weight and size savings.—A. M. F.

How and Where to Use 430 Stainless. R. A. Lincoln and T. A. Pruger. (*Iron Age*, 1953, **171**, Feb. 26, 127–131; Mar. 5, 178–181). The problems involved in replacing Cr–Ni steels by straight chromium steels are discussed. The best opportunities for substitution are where corrosion resistance is of prime importance and other special mechanical properties are not needed. Where welding is necessary, a simple heat-treatment at 1450° F. corrects the lower corrosion resistance of welded areas. Titanium added to type 430 stainless steel improves the ductility of welds and a free-machining grade type 430F closely approaches mild steel in machinability. Toughness at sub-zero temperatures and high strength at high temperatures are not obtained and stainless steel containing nickel must be used for these applications.—A. M. F.

New Austenitic Stainless Good Alternate for 18-8. R. L. Hatschek. (*Iron Age*, 1953, **171**, Mar. 12, 135–138). A new stainless to replace type 301 has been developed by the Budd Co., Philadelphia. This has an analysis of 15·0% Cr min., 16·5% Mn max., 0·1% C max., and 1·0% Ni max. It is being produced by a number of steel companies and is a structural material for trailers and railway coaches. Formability, strength, fatigue and corrosion resistance conform closely with type 301 and no changes in fabrication methods or tooling are required.—A. M. F.

Symposium on the Simplification of Stainless Steel Specifications. (*Trans. Inst. Chem. Eng.*, 1952, **30**, 3, 151–188). This symposium, arranged by the Institution of Chemical Engineers, was held in London on April 1, 1952. Abstracts of the papers follow:

The Simplification of Specifications for Stainless Steels. J. L. Sweeten. (151–155). A group of specifications ranging from 13%-Cr to 25/20 Cr–Ni stainless steel was drawn up and circulated to contributors. Proposed specifications for 18/8 and 18/8 stabilized stainless steels, for 13% Cr and 18/8 Cr–Ni–Mo stainless steels are given. It is finally proposed that British Standards 1449, 1607, 1630–1632 and 1750 be withdrawn, and that B.S. 1501–1508 be simplified and modified.—L. J. L.

The View of the Steel Maker. E. Smith. (156). The author holds that the proposed specifications under discussion do not constitute improvements on existing specifications—a standing technical committee of plant users, plant fabricators, and suppliers of construction materials to consider general problems and to discuss suitable action could achieve more than would be achieved by reliance upon specifications.—L. J. L.

The Foundryman's Point of View. M. M. Hallett and C. Hand. (157–158). The number of specifications used should be kept at a minimum. Individual specifications involved are discussed in detail. The authors comment that the specifications show no advantage over those laid down in British Standards 1630–1632.—L. J. L.

The View of the Semi-finished Product Maker. J. W. Jenkin. (159–161). Every specification should be as wide as possible to serve the maximum number of users. The following criticisms of existing specifications are offered: (1) Martensitic: the carbon and the silicon contents are too high for cold working. (2) Ferritic: maximum carbon content in lowest chromium steel is 0·08%—this tends to restrict availability. (3) The manganese content of higher chromium steel composition is 2%—what advantage is gained by exceeding 1%? (4) Austenitic: the ranges of nickel and chromium are unduly narrow.—L. J. L.

The Fabricator's Viewpoint. J. F. Lancaster. (165–167). The proposed specifications are examined from the points of view of weldability, cold and hot working, and machinability.—L. J. L.

The View of the User. F. H. Keating and W. D. Clarke. (168–169). Consideration should be given to the inclusion of extra low carbon grades. Importance of stabilization is stressed. Titanium is preferred in a standard steel, niobium being reserved for special cases.—L. J. L.

Substitute Stainless Steels. G. A. Dummett. (170–172). The author finds that there is no direct substitute for the austenitic stainless steels for all forms and duties. Within their limits, however, the ferritic stainless irons and mixed structure steels could replace them, if required, for a number of duties with marked saving in raw materials.—R. A. R.

Stainless Steel in Chemical Plant. A. M. Hutcheson. (176–186). The author reviews the metallurgy of stainless steel and argues that shortage of stainless steel may lead to economies in cost of plant. He discusses the more important considerations in the design of units of chemical plant, and enumerates the advantages and limitations of stainless clad material.—L. J. L.

Engineering Data on Wrought 35 Ni-15 Cr Alloy. P. Goetcheus. (*Indust. Heating*, 1953, **20**, Mar., 454–464). Wrought alloys are used to make structures that would not be practical to cast. The 35/15 Ni–Cr alloy is considered in detail. Tables showing its engineering properties are given and heat-treatment, welding, machining, forming, and forging are considered. The alloy is useful at operating temperatures of 1500–2150°F. and is resistant to carburizing and carbonitriding atmospheres in the 1600–1750° F. range. It also has good resistance to thermal shock.—B. G. B.

Selecting Forging Alloys. L. Spencer. (*Product Eng.*, 1953, **24**, May, 202–207). Carbon, medium alloy, high alloy, and high-temperature alloy steels are discussed, and the heat-treatment required and the characteristic properties developed are given. The Amer. Iron Steel Inst. types of commercial forging steel available are listed and grouped with regard to their mechanical properties.—A. M. F.

Heat Treatment and Selection of Steels for Plastic Dies. J. A. Barke. (Plastics Inst.: *Engineer and Foundryman*, 1953, **17**, Mar., 77). Properties of die steels for plastic moulding are discussed. An account is given of the "Firth Brown" range of die steels. General rules are given for heat-treatment of dies.—A. P. H.

Some Notes on Standard Specifications for Rails and Fishplates. A. M. Sims. (*Rail Steel Topics*, **1**, 3, 12–16). The inadequacy of present specifications for rails and fishplates is pointed out. A plea is made for more co-operation between manufacturers and users in the production of rails with longer life.—A. D. H.

Stainless Steels for the Railway Engineer. P. M. Slater. (*Rail Steel Topics*, **1,** 3, 17–23). The author gives a review of the properties and applications of Cr, Cr–Ni, and Cr–Ni–Mo stainless steels for railway engineering.—A. D. H.

A New Steel for Colliery and Railway Draw Gear. A. G. Snape. (*Rail Steel Topics*, **1,** 3, 27–35). The properties of a steel containing 0·12% C, 1·6% Mn are shown to be suitable for colliery draw gear. Compared with mild steel or wrought iron it has a higher tensile strength, lower liability to strain-age embrittlement, and better corrosion resistance, and the properties are maintained at sub-zero temperatures. Flash welding must be used instead of forge welding.—A. D. H.

The Quest for Increased Rail Life. C. J. Allen. (*Rail Steel Topics*, **1,** 4, 10–24). The life of rails under present conditions is less than that of rails rolled 70 years ago. The reasons for this are analysed. Slower rolling speeds and the lighter loads carried are considered to be the main factors contributing to the longer life. The author considers that the limits of composition laid down in the British Standards are too narrow. He suggests : (1) A lower limit of 1% Mn should be specified ; (2) the carbon content should be lower ; (3) Higher sulphur and phosphorus should be permitted ; and (4) the steelmaker should be given more freedom with respect to composition, provided that the material satisfies certain mechanical tests.

High-Tensile Steels in the Construction of Rolling Stock. H. Herbiet and C. Scailteur. (*Ossature Métallique*, 1953, **18,** Mar., 170–177). Applications of low alloy Mn–Cu and Cu–Cr–Ni steels in the construction of lorries, buses, trams, mining cars, and barges are discussed and illustrated.—P. F.

The Ductility of Ferritic Nodular Iron. G. N. J. Gilbert. (*Brit. C. I. Res. Assoc. J. Res. Dev.*, 1953, **4,** Feb., 458–478). The results of an investigation into the influence of silicon and phosphorus on the impact properties of ferritic nodular irons are reported. The material becomes notch brittle when the silicon content exceeds 2·4%—although the elongation in the tensile test remains satisfactory. This latter property also falls to a very low value when the silicon content exceeds 4·5%. Phosphorus also has an embrittling effect.—B. G. B.

Ductile Iron : Watch Copper Build-Up. J. C. Neemes. (*Iron Age*, 1953, **171,** Feb. 5, 162–164). Small amounts of copper increase the strength and hardness and reduce the ductility of ductile iron in the as-cast condition. This is a disadvantage in castings which must comply with minimum elongation or optimum machining requirements, but it helps where strength and wear resistance are needed. Annealing corrects deficiencies in ductility but annealing times must be longer. Care should be taken to guard against copper arising from copper-bearing magnesium additions and returned scrap.

METALLOGRAPHY

Electron Microscopic Study of Worn Surfaces. K. Ogawa and N. Takahashi. (*Métaux–Corrosion–Indust.*, 1953, **28,** Feb., 69–74). The results of a series of experiments carried out on samples of steels which were case-hardened under various conditions and subjected to wear tests are described. Replicas of the surface structure of the samples were examined with an electron microscope. The surface structure of samples carburized in liquid baths is shown to be different from those pack carburized. The change in surface structure due to fusion during wear is clearly shown in photographs.—B. G. B.

Examination of Metal Surfaces by the Electron Microscope. Pt. II—Applications to Metallurgical Research. D. A. Beekhuis and J. A. Shuchmann. (*Metalen*, 1952, **7,** Dec. 31, 444–452). [In Dutch]. The author briefly reviews the examination of graphite precipitation in cast iron and the structures of patented wire and Cr–Mo steel after creep testing.—R. S.

Measurement of Particle Sizes in Opaque Bodies. R. L. Fullman. (*Trans. Amer. Inst. Min. Met. Eng.*, 1953, **197** : *J. Met.*, 1953, **5,** Mar., 447–452). A description is given of methods for the analysis of particle size measurements by area and lineal analysis when the particles are in the form of spheres, plates, or rods. The methods permit description of the distribution of phases present in commonly encountered metallurgical samples.—G. F.

On a Micrographical Method of Studying the Structure of γ-Iron by the Formation of Epitaxial Oxide Nuclei on Its Surface. J. Bardolle. (*Compt. Rend.*, 1953, **235,** May 4, 1790–1791). Controlled oxidation at low pressures produces oxide nuclei on the surface which reveal not only the underlying grain boundaries but also the structure and orientation of individual grains. The technique is being used to study the $\alpha \rightleftarrows \gamma$ transformation during heating and cooling.—A. G.

A New Etching Treatment for the Cementite and Austenite γ-Phase, Especially Deposits of these in the Hardened Structure. H. Klemm. (*Metallkundliche Berichte*, 1951, No. 2 : Verlag Technik). A description of the chemical effect of sodium thiosulphate, used as a metallographical etching medium, upon cementite and austenite is given. Alterations in texture are discussed and illustrated.—L. J. L.

Sodium Thiosulphate Solution, a New Etching Agent to Determine Phosphorus Distribution in Commercial Iron. H. Klemm. (*Metallkundliche Berichte*, 1951, No. 5 : Verlag Technik). A slightly acid, saturated sodium thiosulphate solution was used as etching medium for phosphorus segregation. The results are illustrated in 14 photographs and evaluated.

Autoradiographic Study of the Homogenization of Dendritic Segregations of Phosphorus and Arsenic in Steels. A. Kohn. (*Rev. Mét.*, 1953, **50,** Feb., 139–151). For the ingots examined the effects of high temperature annealing for long periods were : (1) Arsenic exhibits more segregation than phosphorus and is harder to homogenize ; (2) the presence of normal amounts of alloying elements does not affect speed of homogenization ; (3) annealing for 32 hr. at 1200° C. causes marked but not complete homogenization. It is concluded that heterogenities revealed by the classic nital attack correspond well with the actual chemical heterogeneities.—A. G.

More on Hazardous Etchants. D. Warren. (*Metal Progress*, 1952, **62,** Dec., 122). Etchant containing 20–30 c.c. HCl, 10 c.c. HNO_3, and 30 c.c. glycerol should be discarded promptly after use and never stored in stoppered containers.

Identification of the Precipitate Accompanying 885° F. Embrittlement in Chromium Steels. R. M. Fisher, E. J. Dulis, and K. G. Carroll. (*Trans. Amer. Inst. Min. Met. Eng.*, 1953, **197** ; *J. Met.*, 1953, **5,** May, Section 2, 690–695). Results are given of an investigation of the chromium-rich precipitate associated with the embrittlement at 475° C. of high-chromium ferritic steels, the experimental procedure involving optical and electron microscope studies, X-ray examination, chemical analysis of residues, and magnetic measurements. The precipitate is found to have the body-centred cubic structure with lattice parameter equal to that of a pure 70%-Cr 30%-Fe alloy. The characteristics of the precipitate are discussed.

Accidental Hyper-Graphitization. C. H. Junge. (*Metal Progress*, 1952, **62,** Dec., 120, 122). An iron nut which was accidently imbedded in a lump of graphite during manufacture was examined. Proeutectoid graphite flakes grew to tremendous size and photomicrographs of the structure are shown.—B. G. B.

Statistical Grain Structure Studies: Plane Distribution Curves of Regular Polyhedrons. F. C. Hull and W. J. Houk. (*Trans. Amer. Inst. Min. Met. Eng.*, 1953, **197:** *J. Met.*, 1953, **5,** Apr., 565–572). Plane distribution curve have been determined for random sections through four regular polyhedrons, approximating to the shape of metal grains, in order to clarify interpretations of grains structures and to assist in calculations of spatial grain-size distributions. The characteristics of the distribution curve for the average metal grain are predicted.—G. F.

Influence of Manganese on the Diffusion of Carbon in Austenite. M. E. Blanter. (*Zhurnal Tekhnicheskoi Fiziki*, 1951, **21,** 7, 818–821). The influence of the manganese and carbon contents and of temperature on the coefficient of diffusion of carbon in manganese-alloyed austenitic steel was studied. Formulæ for the diffusion coefficient for manganese concentrations of 0·15%, 1%, 12% and 19% are given.—V. G.

The Origin of Specimen Contamination in the Electron Microscope. A. E. Ennos. (*Brit. J. App. Phys.*, 1953, **4,** Apr., 101–106). Contamination of the irradiated surface in an electron beam is shown to originate from the interaction of electrons with organic molecules adsorbed from the vapour phase. Heating the bombarded surface or surrounding it with a cold trap prevents contamination.—J. O. L.

Formation and Properties of Delta-Iron (Ferrite) and Sigma Phase in Austenitic Chromium-Nickel Steels. H. Buchholtz, H. Krächter, and F. Kraemer. (*Arch. Eisenhüttenwesen*, 1953, **24,** Mar.–Apr., 113–125). Tests have shown that austenitic 18/9 Cr–Ni steels do not tend, under normal working conditions, to precipitate ferrite. Stabilizing elements should not be present in too large amounts ; excess titanium,

tantalum, niobium, and molybdenum favour ferrite formation, but even with molybdenum contents up to 2·5% the tendency to ferrite precipitation can be reduced, even at heating temperatures up to 1300° C., by lowering the chromium and raising the nickel contents. The rates of the reversible δ-iron/austenite change depend on the composition of the alloy. The conditions under which the delta can transform to sigma phase have been studied and the results are discussed. Sigma phase causes an increase in hardness and brittleness of these steels. X-ray examination showed that the sigma phase probably has a tetragonal lattice (a = 6·26 Å ; c/a = 1·46). Cold work increases the rate and extent of the transformation of delta to sigma phase.—J. P.

Contribution to the Study of Band Structure in Forged Steel. Combined Influence of Minor Segregation, Migration of Carbon, and Thermal Treatment. P. Cattier, Ch. Dubois, J. Bleton, and P. Bastien. (*Rev. Mét.*, 1953, **50**, Apr., 275–290). The distribution of inclusions is found to depend on the varying hardness of the bands caused by carbon diffusion. In forged parts, significant local heterogeneity persists, even after many reheatings. Carbon diffusion is influenced by the minor segregation of phosphorous and the speed of cooling after austenitizing, the effect of phosphorous being due to its causing the A_3 point to rise and the hardenability to increase.

Screw Dislocations as Strain Embryos of Martensite with the (225)-Habit. O. Krisement. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar.–Apr., 191–198). Experimental observations and calculations by the generally accepted formulæ for the stress field of a screw dislocation show clearly that screw dislocations belonging to the normal $\{111\} - \langle 110 \rangle$ slip systems of austenite fulfil the conditions for a strain embryo of martensite of (225)-habit and the Kurdjumow-Sachs orientation relationship.—J. P.

Study of the Surface Tension of a Grain Boundary in a Metal as a Function of the Orientation of the Two Grains which the Boundary Separates. J. Friedel, D. B. Cullity, and G. Crussard. (*Acta Metallurgica*, 1953, **1**, Jan., 79–92). [In French]. The interfacial tension between two grains of an aluminium disc is measured by the classical method of the three-way corner. The results show that a boundary between two grains whose cyrstal lattices have a common plane of symmetry has a lower interfacial tension which is further decreased if the plane of the boundary is the plane of symmetry. The interfacial tension is almost zero when the grains are parallel or in spinel twin orientation. A new method of calculating the approximate interfacial energy is given. Values are calculated for α- and γ-iron.—A. D. H.

Complicated Domain Patterns on Iron-Silicon Single Crystals. C. F. Ying, S. L. Levy, and R. Truell. (*J. App. Phys.*, 1952, **23**, Dec., 1339–1345). Stages in the formation of complicated domain patterns on the (100) surfaces of iron-silicon single crystals during electrolytic or mechanical polishing and of their disintegration into simple patterns have been studied. The existence of patterns is considered to be the result of local strain.—J. O. L.

Nucleus Formation in Transformations in the Irreversible Iron-Nickel Alloys. G. Masing and O. Nickel. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar.–Apr. 143–151). The literature of the α-γ and γ-α transformations in the irreversible iron–nickel alloys is discussed, particular stress being laid on thermodynamic calculations and investigations involving prolonged annealing. The transformation diagram of iron–nickel alloys has been determined by thermal analysis and differences as compared with previous work discovered. The composition of phases precipitated during transformation does not correspond to equilibrium. An explanation for this is offered, and similar phenomena in Fe–Mo, Ni–Si and Au-Ni alloys are described.—J. P.

Undercooling in Transformation Processes as a Basis for the Martensite Transformation. E. Houdremont and O. Krisement. (*Arch. Eisenhüttenwesen*, 1953, **24**, Jan.–Feb., 53–57). Polymorphic changes in pure materials which involve no mechanical work occur at a temperature at which the two phases have equal free energies. When mechanical work is performed as a result of the transformation, the latter exhibits a temperature hysteresis. By making some simplifying assumptions, an expression is derived for the mechanical work required to form nuclei and this in turn is related, thermodynamically, to the temperature hysteresis. The results agree well with the experimentally determined values of the hysteresis at the A_3 point in pure iron. These considerations also apply to transformations in alloys, provided that no concentration changes occur. The martensite transformation in iron–carbon alloys fulfills this conditions and it can be considered as an undercooled diffusionless γ–α transformation, the extent of undercooling depending on the amount of mechanical work. Calculated results are of the same order as those found experimentally. Comparison of the lattice constants of residual austenite with those obtained by extrapolation to zero alloy content of results on alloyed austenitic steels allows conclusions to be drawn as to the amount of residual austenite and the stress therein as a function of the carbon content. The applicability of these considerations of the γ–α transformation in iron-nickel alloys and the transformation stages in iron–carbon alloys is discussed.—J. P.

Abnormal Structure in Alloys of Eutectic or Eutectoid Type (4th Report). Abnormal Structure in Ternary Eutectic. K. Nagasaki. (*Nippon Kinzoku Gakkai-Si*, 1952, **16**, Jan., 23–26). [In Japanese]. Structures of eutectic alloys in ternary and binary systems are discussed in terms of variations in proportions, and are illustrated from the systems Cd–Bi–Sn, Al–Cu–Si, Al–Cu–Mg, and Fe–Fe_3C–Fe_3P.—K. E. J.

Thermal Stabilization of Austenite in Iron–Carbon–Nickel Alloys. E. R. Morgan and T. Ko. (*Acta Metallurgica*, 1953, **1**, Jan., 36–48). [In English]. Using alloys containing 0·9–1·3% C and 0–10% Ni, it is shown that the M_s point is depressed by stabilization above this point. This effect is reflected in an increase in the amount of retained austenite on slow quenching to a reference temperature. The amount of martensite lost during stabilization is recovered by further cooling. The deficiencies in previous theories of thermal stabilization are pointed out and a mechanism of stabilization based on generation of a Cottrell atmosphere around dislocations is suggested to account for the increase in resistance to martensite formation.—A. D. H.

Martensite. F. C. Frank. (*Acta Metallurgica*, 1953, **1**, Jan., 15–21). [In English]. Martensite is considered as having a face-centred tetragonal unit cell, assuming that each close packed plane of the α-phase meets a close-packed plane of the γ-phase, that in the Kurdjumov Sachs case this meeting occurs in close-packed rows, the lattice plane and habit plane may be calculated using the known lattice parameters. Agreement with observation is within experimental error. It follows that a grid of screw dislocations must be present parallel to the close packed rows.—A. D. H.

Decomposition of Austenite in Low Carbon, Ni, Mo, Steel Weld Metal. E. F. Bailey and W. J. Harris, jun. (*Welding J.*, 1953, **32**, Mar., 134s–138s). The isothermal decomposition of steel weld metal (0·08% C, 0·45% Si, 1·05% Mn, 2·00% Ni, 0·75% Mo, 0·25% N_2, 0·25% Cr) was studied. The TTT diagram showed three reaction ranges : (1) Upper and intermediate types of nucleation ; (2) growth reactions with no eutectoid microstructure ; and (3) a martensite reaction at lower temperatures. The influence of decomposition during cooling to an isotherm on isothermal decomposition is discussed. A completely martensitic structure was obtained only at water-quenching rates.—U. E.

Application of Nucleation Theory to Isothermal Martensite. J. C. Fisher. (*Acta Metallurgica*, 1953, **1**, Jan., 32–35). [In English]. An extension of classical nucleation theory is applied to the martensite transformation. Previous data obtained with a 23%-Ni, 4% -Mn, 73%-Fe alloy for isothermal transformation of martensite agree well with the theory.

The Iron/Iron-Sulphide/Cobalt-Sulphide/Cobalt Phase Diagram. R. Vogel and G. F. Hillner. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar./Apr., 133–141). The above field of the Fe–S–Co system has been studied by cooling curve, microscopical, X-ray, and chemical analysis methods. The FeS–CoS system forms a continuous solid solution, the lattice parameter changing regularly with composition. The primary precipitation surface of the three metallic phases, δ-(FeCo), γ-(FeCo) and α-(FeCo) and the sulphides η-(Fe)S and β-Co_4S_3 were determined. The equilibrium relationships in the quaternary system were studied on sections at a number of Fe : Co ratios. The compositions of solid solutions in equilibrium with each other indicate that sulphur has a stronger affinity for iron than for cobalt. It is to be noted that at a cobalt content of 12·5%, sulphur has an appreciable solid solubility (approximately 1% at 1450° C.).—J. P.

The Irreversibility of Iron-Nickel Alloys. E. Scheil. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar.–Apr., 153–160). The con-

cept of irreversibility and the limiting types of transformation, thermal and athermal, are discussed. In irreversible transformations there is no equilibrium because the change does not occur by thermal interchange of atoms but by processes analogous to those of twin and slip plane formation. The thermodynamics of irreversibility are briefly examined.—J. P.

CORROSION

The Passivity of Iron. K. F. Bonhoeffer. (*Z. Metallkunde*, 1953, **44**, Mar., 77–81). A survey is made of the present state of knowledge of passivation and activation phenomena exhibited by iron in acid solutions. After describing various phenomena of passivation and re-activation produced chemically and by anodic treatment, the author discusses the work done in interpreting the experimental results, and summarizes the conclusions obtained. (11 references).—L. D. H.

The Passivity of Iron. T. G. O. Berg. (*Z. Metallkunde*, 1953, **44**, Mar., 82–83). The author briefly summarizes some of his previously published results relating to passivation and activation of iron, copper, and iron-chromium alloys, and the measurement of solution velocities and potentials in nitric acid of varying concentration.—L. D. H.

Polarization and Passivation in Metallic Corrosion. W. Katz. (*Z. Metallkunde*, 1953, **44**, Mar., 108–112). By comparing current-potential curves with potential-time curves made either by immersion of the electrodes without current or by polarization, the occurrence and mode of action of corrosion layers are demonstrated. Examples are given of ferrous and non-ferrous metals.—L. D. H.

Practice and Theory of Corrosion in a Chemical Plant. M. Werner. (*Z. Metallkunde*, 1953, **44**, Feb., 37–43). Experience in the chemical industry with metallic materials shows that industrial corrosion is not fully explained by the laws of electro-chemical action ; the durability of the material is largely measured by the properties of the protective layers which are formed. Several examples and illustrations are given.—L. D. H.

Corrosion Factors in Chemical Engineering. F. Howard. (1952 Welding Engineering Convention : *Australasian Eng.*, 1952, Dec. 8, 67–71). The mechanisms of corrosion of metals are outlined, including contributory factors. Common tests employed to determine the rate of corrosion and the corrosion resistance of materials used in chemical engineering are outlined. Corrosion associated with the welding of various materials is considered, and the main materials of construction in the chemical industry, including stainless and clad steels are treated in the light of their corrosion resistance. Industrial methods of controlling corrosion, are detailed.

Inhibition of Corrosion. U. R. Evans. (*Chem. Indust.*, 1953, May 30, 530–533). Corrosion, implying the continuous transfer of ions from a metallic surface into a contiguous liquid, cannot proceed in a neutral, oxygen-free medium or in the presence of an oxidant sufficiently active to form a protective film on the metal surface. Since the first condition is commercially unattainable, the second is obtained by adding an inhibitor, either to decrease the surface activity so that the normal supply of dissolved oxygen is sufficient for protection or to increase the oxidising power of the solution. Various types of inhibitor are described, including recently developed organic compounds which appear to act by adsorption on reactive areas.—J. O. L.

The War Against Corrosion. A. G. Thomson. (*Min. J.*, 1953, **240**, Apr. 3, 388–390 ; Apr. 10, 422–424). After reviewing the tremendous damage caused each year by the corrosion of metals, the author discusses a number of methods of protection.—B. G. B.

Economic Aspects of Cathodic Protection. R. M. Wainwright. (*Corrosion*, 1953, **9**, Feb., 51–55). Cathodic protection in the U.S.A. could eliminate nearly one-fifth of present corrosion losses. For pipelines, annual cost of corrosion can be reduced from 2–5% to 1–2% of the original investment. Factors to be taken into account are discussed.—J. F. S.

Protection of Metallic Structures Against Atmospheric Corrosion. D. Bermane. (*Ossature Métallique*, 1953, **18**, June, 333–340). Recent developments in the theory and practice of corrosion protection are surveyed. The influence of different types of atmospheric pollution on the corrosion rate, the mechanism of failure of protective coatings, and the preparation of metallic surfaces for coating are considered, and the choice, quality, and application of paints and undercoats are discussed. Experimental results relating to the efficiency of various types of coating are presented.—P. F.

Application of Electric Analogues to the Study of Diffusion Phenomena in Differential Atmospheric Corrosion. G. Bianchi. (*Metal. Ital.*, 1953, **45**, Apr., 123–127). The author examines the possibility of using electric analogues to study diffusion phenomena, using both direct and conjugate field models. Experimental methods used are reviewed and the results obtained with electric analogues in the study of oxygen diffusion in differential air corrosion are reported. Comparison of these results with those obtained from corrosion tests on zinc confirms the assumption that the oxygen concentration is nil in the layer of solution which wets the metal.

Protection of Underground Cable Sheaths. T. W. Alexander, jun. (*Iron Steel Eng.*, 1953, **30**, Apr., 97–100). The author examines briefly the various types of corrosion to which underground cable sheaths are submitted. The effects of stray current anodic and cathodic corrosion are discussed as well as the effects of localized anodic corrosion, galvanic action and chemical attack. Remedial measures are suggested.

Prevention of Corrosion on Underground Pipeline. (*Commonwealth Eng.*, 1953, **40**, Feb. 2, 294–295). A description is given of the physical and cathodic corrosion protection techniques to be applied on the Shell Company's pipeline which will transport oil refinery products in Australia.—L. J. L.

Protective Coatings for Buried Pipelines. K. A. Spencer and H. B. Footner. (*Chem. Indust.*, 1953, May 9, 448–455). Economic as well as practical reasons justify the continued use of pitch and bitumen coatings for buried pipelines, differentiation being at present a matter of individual preference. Coatings of $\frac{3}{16}$ to $\frac{3}{8}$ in. thickness over wrapped pipe are preferable, but with cathodic protection, $\frac{3}{32}$ to $\frac{1}{5}$ in. coatings can be used on long runs with reasonable success. Tests can be used both before and after installation to confirm the efficiency of the protective coating.—J. O. L.

Corrosion Inhibitors in Neutral Aqueous Solutions. F. Wormwell. (*Chem. Indust.*, 1953, June 6, 556–561). Work at the Chemical Research Laboratory has shown that certain organic sodium salts inhibit the corrosion of cast iron, soldered joints or zinc. The tendency for more even corrosion in event of failure of these inhibitors is explained and conclusions are drawn on the mechanism of the protective action. Fundamental investigations, using electrode-potential and radioactive-tracer techniques, are proceeding.—J. O. L.

On the Solution Potential of Ferro-Chromium Alloys after Abrasion in an Inert Atmosphere. H. Hatwell. (*Compt. Rend.*, 1953, **256**, May 11, 1881–1883). With a continuously abraded surface the potential is independent of the chromium content and close to the value for pure iron. This demonstrates that passivity is not due to the high chromium content itself, but to the presence of a protective layer.—A. G.

Cavitation Laboratory for Hydraulic Turbines. H. Ulmann and R. S. Sproule. (*Mech. Eng.*, 1953, **75**, Mar., 194–198). Details are given of test equipment and calibration procedure in a new cavitation laboratory of the Dominion Engineering Works, Montreal.—D. H.

Cavitation. E. Crewdson. (*Engineer*, 1953, **195**, Jan. 23, 122–123). The author suggests that the damage done by cavitation is not the consequence of the collapse of bubbles, and maintains that such collapse cannot take place. The high pressure and the high temperature of the gas within the bubble can account for observed damage under cavitation conditions.—M. D. J. B.

Corrosion and Marine Pitting. B. Callame and H. Rabaté. (*Peintures, Pigments, Vernis*, 1953, **29**, Feb., 115–121). In an address given in memory of C. Francis-Boeuf, details of the organization and activities of the Centre for Oceanographic Researches and Studies at La Rochelle-Pallice are presented.

Theory of Scaling of Metal Alloys. K. Hauffe. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar.–Apr., 161–171). The rate of scaling of metals at high temperatures by the action of oxygen, sulphur, and halogens is controlled by diffusion processes in the compact scale. It is shown that Wagner's lattice defect theory for pure metals and its extension to alloys by Wagner, and by Hauffe and co-workers, can explain the scaling rates of iron, copper, nickel, silver, titanium and zinc, and their alloys, provided that the process is not accompanied by phase boundary reactions and diffusion through pores as a result of faults in the scale. Some indications are given for the development of scale-resistant alloys.—J. P.

Contribution to the Study of Oxide Film Growth on Iron at Annealing Temperatures. F. Savioli. (*Metal. Ital.*, 1952, **44**,

Dec., 598–599). The author reviews methods of measuring the thickness of oxide films, by interference colours, electrical, electro-chemical, and optical means. Research carried out on the growth of two grains of ferrite in a low carbon steel shows a linear relation between film thickness and the square root of temperature, thus confirming that the reaction follows a parabolic law. A small furnace fitted with a Leitz microscope was used to follow the colour variation of the grains during oxidation at different temperatures. (18 references).

Correlations between Parabolic Oxidation of Metals and Properties of Oxides. A. Dravnieks. (*J. Electrochem. Soc.*, 1953, **100,** Mar., 95–102). The parabolic rate constants for the oxidation of a number of metals at a standard temperature have been modified, in the light of modern theories, so as to relate to the same number of diffusion paths and jumps, and the same driving force per jump. In the divalent oxides, the modified constant increases with decreasing cation size for the same lattice type, the values for the wurtzite type being smaller than those for the rock salt type. In the tri- and tetra-valent oxides, the constant increases with cation size. The relationships between the modified constant and other properties and thermodynamic functions of the oxides are discussed.—J. P.

Effect of Heat-Treatment and Related Factors on Straight-Chromium Stainless Steels. F. K. Bloom. (*Corrosion*, 1953, **9,** Feb., 56–65). The hardenable chromium stainless steels are increasingly used in the oil and petroleum industry. The effects of heat-treatment and carburization on their mechanical properties and corrosion resistance are described. In many media the corrosion resistance varies with the heat-treatment.

The Corrosion of Domestic Galvanized Hot Water Storage Tanks. I. L. Newell. (*Corrosion*, 1953, **9,** Feb., 46–51). Severe corrosion of galvanized water tanks occurred when traces of copper were present or when the temperature exceeded 140° F. Bicarbonates and/or nitrates also caused attack.—J. F. S.

The Coupon Technique—A Valuable Tool in Corrosion Testing. H. L. Bilhartz and H. E. Greenwell. (*Corrosion*, 1953, **9,** Feb., 66–73). The technique used by an oil and gas producing company to assess corrosion with the aid of specimens cut from coils of shim stock 0·004 and 0·012 in. thick exposed in Bakelite holders, and the exposure record cards used, are described.—R. A. R.

ANALYSIS

The General Motors Research Corrosion Test: A Cyclic Humidity Accelerated Corrosion Test for Sheet Steel. A. J. Opinsky, R. F. Thomson, and A. L. Boegehold. (*Amer. Soc. Test. Mat. Bull.*, 1953, Jan., 47–51). A new test has been developed to study the corrosion resistances of bare steel specimens for those applications where the protective rust of specimens exposed on racks outdoors is not fully realized. A slow humidity cycle, an elevated temperature, and a dip in weak electrolyte, are arranged so that an almost non-protective rust is developed. Corrosion losses of steels exposed to this test are about three times those of the same steels exposed to a modified A.S.T.M. salt-spray test.—B. G. B.

Utility of Strauss Test. L. H. Satz. (*Metal Progress*, 1953, **63,** Jan., 136, 38). The use of $CuSO_4$–H_2SO_4 solutions for detection of the sensitivity of austenitic steels to intergranular corrosion (the Strauss test) is described. Several papers on this test are discussed and suggestions for future work are made.—B. G. B.

Steel Protection by Sprayed Aluminium. (*Engineering*, 1953, **175,** Jan. 30, 158). A steel building, a test house at a Sheffield steelworks, situated in a highly corrosive atmosphere and protected by a sprayed aluminium coating, was inspected after 14 years and only slight rusting found.—J. McK.

Corrosion Problems in Steam-Jet Vacuum Equipment. D. H. Jackson. (*Chem. Eng. Prog.*, 1953, **49,** Feb., 102–104). The severe conditions in the steam nozzle of ejectors are explained. Linings with stainless steel, Monel metal, carbon, Hastelloy, and other materials are mentioned.—T. E. D.

Faulty Firing as Cause of Corrosion in Steam Boilers. K. Schwarz. (*BWK*, 1953, **3,** Mar., 94–96). It is shown on the basis of experimental data that smoke and dust deposits in steam boilers, arising from incomplete combustion, can be traced to faulty firing practice. The deposits accelerate corrosion of the boiler tubes. Precautions necessary to avoid corrosion of this type are indicated.—P. F.

Some Observations on Corrosion in Engineering. S. F. Dorey. (*J. Inst. Petroleum*, 1952, **38,** Nov., 885–918). A detailed examination of a wide variety of corrosion problems which, in particular, occur in the marine engineering and petroleum industries is presented. The mechanisms of the processes are discussed and treatment to alleviate or inhibit the corrosion are described.—B. G. B.

The Preservation of Oil Tanker Hulls. J. Lamb and E. V. Mathias. (*North East Coast Inst. of Eng. and Shipbuilders*, Mar. 30, 1953, Advance Copy). Protection of the outside of the hull is shown to depend for success as much upon the initial preparation of the surface as upon paint. Flame-cleaning has been found extremely useful for preparing the surface and it is beneficial if the primary coat is applied to the warm plating. The difficulty of protecting the interior surfaces of cargo tanks is shown by the results of numerous tests. Experiments with cathodic protection, using magnesium anodes, are being conducted.—B. G. B.

Acid- and Heat-Resisting Properties of Nodular Cast Iron. I. Iitaka and K. Sekiguchi. (*Waseda Univ., Rep. Casting Res. Lab.*, 1952, No. 3, 17–19). [In English]. Comparative results are presented for nodular and commercial irons, and mild steel. Nodular iron has better resistance than normal iron to weakly corroding media, but less to severe media, *e.g.*, NaCl and H_2SO_4 solutions. Its oxidation resistance, and resistance to growth after repeated heating and cooling, are also superior to those of normal iron.—K. E. J.

The Polarographic Determination of Manganese and Iron. J. V. A. Novák, J. Kůta and J. Říha. (*Chemické Listy*, 1953, **47,** 5, 641–648). [In Czech]. Manganese and iron give well defined polarographic waves in an alkaline solution of triethanolamine. This effect is utilized for the determination of both elements in mine waters and iron ores, of which several thousand samples were analysed. The time required for a determination is shorter than with standard practice. A saving in chemicals is also achieved, 200 g. triethanolamine, 500 g. sodium hydroxide, and about 70 g. of zinc being sufficient for 1000 determinations.—P. F.

Contribution to the Study of Electrometric Titration Estimation Methods. G. Duyckaerts. (*Anal. Chim. Acta*, 1953, **8,** Jan., 57–64). A general equation for the polarization curve of an oxidation–reduction system is established, taking into account the kinetics of reactions and concentration polarization. Its application to the interpretation of titration curves obtained by two methods is described.—T. E. D.

Determination and Identification of Inclusions in Steel. J. Massinon. (*Rev. Mét.*, 1953, **50,** Apr., 264–274). The method of alcoholic iodine solution (Rooney) has been compared with the electrolytic method (Kaiser Wilhelm Institut für Eisenforschung) and that employing anodic attack. Reproducible results are attainable for steels with up to 0·55% carbon and silicon below 0·22%. The use of the method in works' laboratories is considered.—A. G.

Contribution to the Determination of Lead in Steels. D. Rosi. (*Metal. Ital.*, 1953, **45,** Jan., 22–23). A quantitative study of the precipitation of lead sulphide shows that small variations of HCl are sufficient to upset the optimum acidity. The author proposes a method based on the precipitation of lead sulphide in acid solution by tartaric acid, and establishes the best conditions for obtaining a quantitative precipitation of PbS. (16 references).—M. D. J. B.

The Automatic Electrical Ignition of the D.C. Arc in Spectrographic Analysis. J. M. Nobbs. (*Brit. J. App. Phys.*, 1953, **4,** Apr., 118). A circuit diagram is given for electrically igniting a spectrographic arc by a high-frequency trigger spark, which is adjusted to re-ignite the arc if the latter is weakened or extinguished.—J. O. L.

Polarographic Determination of Tin in Tin Plating Solutions. R. Diaz. (*Plating*, 1953, **40,** Jan., 45–46). Tin in stannate plating baths is determined polarographically in 1·6*N* perchloric acid and 0·8*N* sodium chloride, using gelatine as maximum suppressor.—J. P.

The Determination of Boric Acid in Nickel Plating Baths. E. J. Serfass, R. B. Freeman, and E. Pritchard. (*Plating*, 1953, **40,** Jan., 59–62). Boric acid is determined by alkali titration in presence of mannitol and of potassium ferrocyanide to remove the nickel.—J. P.

A Note on the Effect of Hydrofluoric Acid in the Determination of Graphitic Carbon. (*Brit. C. I. Res. Assoc. J. Res. Dev.*, 1953, **4,** Apr., 520–521). Nitric acid is used to decompose the sample and to dissolve the carbides in cast irons, leaving

the graphite unattacked. A small amount of hydrofluoric acid hastens the filtration by decomposing any silicic acid formed. This use of hydrofluoric acid has no effect on the residual graphite.—B. G. B.

How to Avoid Mixing Up Steels. P. Devilder. (*Mét. Constr. Mécan.*, 1953, **85,** Feb., 95–99). Physical and chemical methods of separating and identifying different steels which have been mixed are discussed. The former include hardness, magnetic and spark tests, and the latter spot tests for nickel, chromium, manganese, molybdenum, silicon and aluminium.

Sulphur Determination in Steel and Grey Iron by the Combustion Method. F. Auerbach. (*Met. u. Giesserei Techn.*, 1952, **2,** Dec., 433–444). A description is given of an instrument for the rapid determination of the sulphur in steel and grey cast iron by oxidation. The apparatus consists of an automatic pressure valve for the oxygen supply, a nitration chamber for determining sulphur, and a burette for the determination of carbon. Reasons are given for inaccurate values obtained with other instruments.—L. J. L.

Practical Notes on Gas Analysis Using the Orsat Apparatus. P. Chandelon. (*Techn. Human*, 1952–53, **50,** 3, 134–165). The author discusses the procedure to be adopted in using the Orsat apparatus and notes errors which may occur. The importance of correct technique in taking samples is emphasized. The efficacy of the absorbent solutions used is considered and for the absorption of CO the use of finely-divided cuprous oxide in concentrated sulphuric acid is advised instead of, or in addition to, the usual cuprous chloride solutions.—J. McK.

Polarographic Methods in the Iron and Steel Industry. Part III—Polarographic Analysis of Steels. F. Burriel Marti and M. Cruz Serna. (*Inst. Hierro Acero*, 1953, **6,** Jan.–Mar., 29–36). [In Spanish]. Spanish research into polarographic methods in the analysis of iron and steel is reviewed. The interference of iron is explained and notes on the determination of copper, nickel, cobalt, manganese, chromium, molybdenum, wolfram, vanadium, lead, titanium, tin, aluminium and arsenic are given. (45 references).—R. S.

Determination of Small Percentages of Boron in Heat-Resisting Alloys and Alloy Steels. A. G. Komarovskii. (*Zavodskaya Laboratoriya*, 1950, **16,** 10, 1228–1230). A spectrographic method for determining boron in carbon and alloy steels is described. The average error was within $\pm$ 2·8%. The influence of other components on the intensity ratio of the spectrum was examined.—V. G.

Quantitative Analysis by X-Ray Fluorescence. E. Gillam. (*Metal Treatment and Drop Forg.*, 1953, **20,** Mar., 99–102). The application of quantitative analysis by fluorescent X-rays to binary and ternary alloy systems is described. The secondary fluorescent X-rays, emitted by the elements under examination, are analysed by a fluorite crystal on a Geiger counter spectrometer. The difficulties of interpretation of the results, caused by absorption and mutual fluorescence, are discussed. With the usual powerful X-ray intensities, sufficiently accurate results can be obtained in a few minutes, but only if at least 10% of the element is present. Higher accuracies should be possible with more powerful equipment.

Application of the High Frequency Titration. II—The Precipitation Analysis of Small Amounts of Sulphate and Its Application to the Determination of Sulphur in Iron and Steels. S. Musha. (*Sci. Rep. Res. Inst. Tôhuku Univ.*, 1952, **A4,** Dec., 575–581). [In English]. The amount detectable is 1–5 mg. per 100 ml. of sulphate ion using $BaCl_2$ solution in a methanol medium. Satisfactory results were obtained for rapid determinations of sulphur in iron and steels (analytical error approx. 2%).—K. E. J.

The Sampling of Pig and Cast Iron for Carbon Determination. W. E. Clarke. (*Brit. C. I. Res. Assoc. J. Res. Dev.*, 1953, **4,** Apr., 496–519). The results of a co-operative investigation into the best method of sampling pig and cast irons for carbon determination are given and discussed ; only flake graphite materials were examined. The difficulties in obtaining a representative sample depend not so much on the total carbon content as on the size and distribution of the graphite. For very accurate work, solid samples should be used. Satisfactory results are obtained with drillings for machine-cast pig irons and castings less than 1 in. thick.—B. G. B.

Silicate Analysis by a Rapid Semi-Microchemical System. R. B. Corey and M. L. Jackson. (*Analy. Chem.*, 1953, **25,** Apr., 624–628). An analytical system is described utilizing rapid semi-microchemical methods. Two 0·1-g. samples are required, one being fused with sodium carbonate, the other being volatilized with hydrofluoric acid. Silicon, aluminium, iron, titanium, calcium, magnesium, sodium, and potassium can be determined in eight materials in one day.—A. G.

Electrochemical Method for Oxygen Determination in Gases. M. G. Jacobson. (*Analy. Chem.*, 1953, **25,** Apr., 586–591). From results obtained in a study of the depolarization by gaseous oxygen of a carbon cathode polarized in a galvanic cell, a portable and a continuous indicator for oxygen in gaseous mixtures have been developed. The continuous indicator is described in detail ; it is unaffected by CO and up to 24% CO_2 and has an accuracy of 2% of the full-scale range.—A. G.

The Determination of Pyritic Sulphur in Solid Fuels. W. Radmacher and P. Mohrhauer. (*Glückauf*, 1953, **89,** May 23, 503–511). A reduction process is described for the determination of pyritic sulphur, in which the presence of $Cr^{\prime\prime}$ ions. A very fine state of subdivision of the materials contributes to the speed and accuracy of the method. Organically combined sulphur is not reduced. Modification of Powell and Parr's oxidation method are described, giving a reduced time of reaction.—L. D. H.

HISTORICAL

The History of the Black Country Iron Trade. W. K. V. Gale. (*Iron Coal Trades Rev.*, 1953, **166,** Apr. 3, 767–771). The author reviews the technical history of the Black Country iron trade and corrects some errors and omissions occurring in other historical surveys of this area. The reasons for the growth of this iron trade and its subsequent decline are discussed.—G. F.

Early History of the Manufacture of Pig Iron in Spain. J. Almunia y de Leon. (*Inst. Hierro Acero*, 1953, **6,** Jan.–Mar., 43–52). [In Spanish]. The author traces the development of the bloomery hearth from the XIVth century in various parts of the world. The first mention of pig iron in Spain was about 1485 in the chronicles of Hernando del Pulgar. A bibliography is given relating to early developments in Spain. (18 references).—R. S.

Non-Destructive Testing of Ancient Metallic Objects. G. Ambrosino and P. Pindrus. (*Rev. Mét.*, 1953, **50,** Feb., 136–138).

ECONOMICS AND STATISTICS

The British Iron and Steel Industry in 1952. (*Engineer*, 1953, **195,** Jan. 2, 29–31 ; Jan. 9, 65–68). A detailed review is given of all major development schemes in the iron and steel industry in 1952.—M. D. J. B.

Metallurgical Developments in 1952. (*Engineer*, 1953, **195,** Jan. 16, 109–110). A brief description is given of metallurgical developments in 1952 with particular emphasis on economy and conservation of metals.—M. D. J. B.

Yugoslavia's Expanding Mining Industry. A. G. Thomson. (*Min. J.*, 1953, **240,** Mar. 20, 328–331). Details are given of the production of chromite and magnesite and of the development of the steel and coal industries of Yugoslavia.—B. G. B.

The European Steel Industry and the Wide-Strip Mill. (*United Nations Dept. of Economic Affairs, Industry Division, Economic Commission for Europe*, Geneva, 1953). This 100-page book is a study of production and consumption trends in flat products.—R. A. R.

Present Aspects of the Iron and Steel Industry in the United States. A. G. Lefebvre. (*Pub. Ass. Ing. Faculté Polytechn. Mons.*, 1952, No. 2, 1–44). A comprehensive survey of the U.S. iron and steel industry is given. All aspects are considered, from the mining of the ore to the finished products.

Alloying Metals in Special Steels, and Their Supply. A. Michel. (*SIM Documentation Métallurgique*, 1952, Oct.–Dec., 133–140). The use of chromium, nickel, tungsten, molybdenum, vanadium and cobalt are discussed. Consumption of these metals in France is listed and their annual productions in various parts of the world are surveyed. Variations in market prices are reviewed.—T. E. D.

Industrial Power Plant Construction Costs since World War II. T. A. Fearnside and F. C. Cheney. (Amer. Soc. Mech. Eng. : *Blast Furn. Steel Plant*, 1953, **41,** Apr., 420–421, 424–425). The authors derive, after study of the actual project cost of several new power plants, three factors which, when multiplied by the purchase price of the boilers, turbines and condensers, give an idea of the total project cost of an indus-

trial power plant. The boiler, turbine and condenser factors are 3·61, 1·69 and 1·86, respectively.—B. G. B.

MISCELLANEOUS

The Principles of Modern Statistical Evaluation Procedures with Reference to Sampling. H. Klein. (*Arch. Eisenhüttenwesen*, 1953, **24**, Jan.–Feb., 11–20). A clear presentation is given of the principles of statistical procedures to enable their value in sampling to be assessed. The possibilities of devising specifications for the number of random samples, and acceptance and rejection on the basis of test results are discussed. It is shown how statistical evaluation procedures applied to random samples can throw light on the accuracy of sampling.

The Usefulness of Mathematical Procedures in Sampling after Elimination of Biassed Errors. H. Jahns. (*Arch. Eisenhüttenwesen*, 1953, **24**, Jan.–Feb., 21–26). Examples are given to illustrate the usefulness of mathematical procedures for indicating the accuracy and suitability of sampling methods. The procedures are applicable only when biassed errors are avoided.—J. P.

New Contributions to International Standards Relating to Flow Measurement. R. Witte. (*BWK*, 1953, **5**, June, 185–190). A report is given of the deliberations and decisions of the first two post-war meetings on International Standards on Flow Measurement, as well as on related experimental work by the author. The design and performance of venturi-type flow meters receives particular attention.—P. F.

Careers in the Iron and Steel Industry—West Riding and North Derbyshire Area. (*Brit. Iron Steel Fed.*, 1953). A very short account is given of the iron and steel industry of the West Riding and North Derbyshire district, and the training and opportunities available to a youth entering the industry are outlined.—R. A. R.

A Career in the Iron and Steel Industry. (*Brit. Iron Steel Fed.*, 1953). This 16-page booklet outlines some of the careers open to a young man entering the iron and steel industry.—R. A. R.

Standard Cost Accounting in the Iron Industry ? H. Diercks and F. Petzold. (*Arch. Eisenhüttenwesen*, 1953, **24**, Mar.–Apr., 183–190). The methods by which standard cost accounting may be applied to the iron-producing industry and the advantages to be gained thereby are discussed on the basis of its use in other industries and of experience in America.—J. P.

Measurement of Productivity—Work Study—Application and Training. (*Joint Commitee of Inst. of Cost and Works Accountants and Inst. of Production Engineers*, 1952).

BOOK NOTICES

American Society for Testing Materials. "1952 *Book of A.S.T.M. Standards, Including Tentatives.*" 8vo. Illustrated. Part 1 : "*Ferrous Metals*," pp. xxviii + 1572. Part 2 : "*Non-Ferrous Metals*," pp. xxix + 1327. Part 3 : "*Cement, Concrete, Ceramics, Thermal Insulation, Road Materials, Waterproofing, Soils*," pp. xxxv + 1634. Part 4: "*Paint, Naval Stores, Wood, Fire Tests, Sandwich Constructions, Building Construction*," pp. xxvii + 1152. Part 5 : "*Fuels, Petroleum, Aromatic Hydrocarbons, Engine Anti-freezes*," pp. xxvi + 1253. Part 6 : "*Rubber, Plastics, Electrical Insulation*," pp. xxx + 1488. Part 7 : "*Textiles, Soap, Water, Paper, Adhesives, Shipping Containers*," pp. xxviii + 1334. Philadelphia, Pa., 1953 : The Society. (Price $76 for seven parts).

Since 1944, A.S.T.M. Specifications have grown from three volumes, stoutish, but containing a mere 5000 pages, to six of 6500 pages in 1946, six of 8200 pages in 1949, to seven, printed on 'Bible paper', which gives them a slimness concealing the 9770 pages of the 1952 edition. Very much of this expansion, however, has taken place outside the metallurgical field, as the titles of the parts will indicate. In 1944 the 'Metals' volume had 2046 pages and contained but 400 specification and methods of testing ; in 1952, the volumes 'Ferrous Metals' and 'Non-Ferrous Metals' total 2900 pages between them and contain 500—not an excessive increase, considering that there are now 1882 A.S.T.M. standards extant. It is difficult to see where, if anywhere, the process of standardization may end, but it is clear that the work done, and being done, by the A.S.T.M. will always be prominent and valuable.—J. P. S.

Institute of Metals. "*Properties of Metallic Surfaces.*" A Symposium organised by The Institute of Metals and held at the Royal Institution, London, on 19 November, 1952. (Institute of Metals Monograph and Report Series, No. 13). 8vo, pp. iv + 368. Illustrated. London, 1953 : The Institute. (Price 35*s*.).

The papers presented at this Symposium can, with a little straining, be divided into three classes which deal respectively with techniques for studying metallic surfaces, with the chemical properties of such surfaces, and with the relations between surface condition and mechanical properties. In the first group, Professor Tolansky's review of optical methods will be greatly valued for the clear and authoritative way it describes the techniques of phase contrast, multiple-beam interferometry, and light-cut methods, which are now becoming so important in metallography. The importance of electron diffraction for studying surfaces is emphasized in a paper by Drs. Gay and Hirsch which deals with the distortion of cleaved calcite surfaces on working, and a new method of investigation, involving the measurement of the electrical resistance of surface layers by means of radio-frequency currents, is reported by Drs. Chambers and Pippard.

On the chemical side, Dr. Simnad describes the use of radio-isotopes to locate the centres of chemical activity on surfaces, and produces evidence in support of the theory (by Hoar and Evans) that passivity in chromate solutions is caused by chromium which enters and seals up pores in the oxide film on the surface of the metal. Dovey, Jenkins and Randle describe industrial processes for treating surfaces by means of diffusion coatings and discuss the chemical reactions by which the coating metal is deposited, while Phillips reports an interesting study of the crystal structure and chemical constitution of the oxide film on aluminium. Finally, Dr. Evans gives a characteristically authoritative and readable account of the ways in which the state of the surface affects the course of corrosion.

The papers dealing with mechanical properties form the largest of these groups. Spear, Robinson and Wolfe correlate information on the nature of machined surfaces, especially their roughness and the extent of the worked layer below them, while Dr. Barwell deals with various kinds of failures in bearings in the presence of liquid and solid lubricants. Contrasting sharply with these technological approaches, Professor Andrade discusses the effect of oxide coats on the yield strength of single crystals of pure metals, and shows that, when the surface is sufficiently clean, work-hardening can become remarkably small, almost zero in fact. In a similar spirit, Professor Gurney presents some interesting arguments and experimental evidence on delayed fracture in glass, especially in relation to the attack of the surface of glass by water and carbon dioxide. Love surveys thoroughly the effects of surface treatment, such as induction hardening, diffusion coating, and shot peening, on the fatigue strength of steel, and the paper by Dr. Bowden and Dr. Tabor gives a most valuable survey of friction between rubbing surfaces, as affected by surface films of various kinds, and includes an excellent discussion of the Beilby layer.

The last 70 pages of the book report the discussion that followed the presentation of these papers. Individual contributions are good, an especially interesting one being Dr. Menter's demonstration of the power of the reflection electron microscope for showing surface details, but the overall result is unsatisfying. The discussion seems to jump randomly from one topic to another, following the personal interests of individual speakers rather than a coherent pattern. Ideally, one imagines that if several different groups of people with diverse approaches—metallurgical, chemical, physical, academic, practical—are brought together they will spontaneously bend to the common task of unifying their viewpoints. But in practice this rarely seems to happen. Rather, each group appears usually to be too preoccupied with its own particular problems to have attention to spare for those of others ; and, through the continual interruption, one by another, of each others' separate debates, there is always a danger that they end up by getting on one another's nerves. But then, it is surely too much to expect a five-part fugue to write itself.—A. H. Cottrell.

THE INSTITUTION OF METALLURGISTS. "*The Metallurgy of the Rarer Metals*". Lectures delivered at the Institution's Refresher Courses, 1952. 8vo., pp. 156. Illustrated. London, 1953 : The Institution. (Price 15*s*. 6*d*.).

This book contains a series of eight lectures delivered at the Sixth Annual Refresher Course held at Ashborne Hill from 31st October to 2nd November, 1952. In these lectures seventeen metals are considered, the author in each case being recognised in this country as an authority on the subject. Having regard to the space available, the occurrence, extraction, physical and chemical properties, and application of each metal are adequately dealt with and the book contains valuable up-to-date information.

In certain cases there appears to be considerable controversy regarding the classification of a metal as "rarer," and it appears that this is not always determined by the relative abundance of the metal in the lithosphere and hydrosphere or the ease with which the ferro-alloy may be produced. Tungsten, molybdenum, chromium, and manganese are therefore grouped as rarer metals because of the difficulty in obtaining the pure metal. Titanium and zirconium, tantalum, and niobium are also included in the rarer group, and it is perhaps surprising that vanadium has been omitted. The other metals considered are uranium and thorium, beryllium, and the platinum group.

The nuclear properties of the radioactive elements are described, and, although information is stated to have been restricted for security reasons, this lecture provides interesting reading. A comparison of the various reduction processes for the production of uranium of high purity incorporates an excellent example of the use of curves relating the standard free energy of formation with temperature. Comprehensive flow sheets and diagrams are given throughout the text, and in this manner the extraction of beryllium is made particularly clear. The physical properties, occurrence, and price of the platinum metals are tabulated, and in this lecture rather more space is devoted to the uses and applications. The metallurgy of titanium and its alloys is fairly complete, but it is rather disappointing that the mechanical properties of zirconium and its alloys have not been given for comparison. The fact that both zirconium and titanium are effective grain refiners in steel has probably been overlooked because aluminium is more commonly used for this purpose. The effect of additions of tungsten, molybdenum, manganese and chromium to steel is briefly discussed, a more detailed account being outside the scope of this book.

Advanced students of metallurgy should find this book particularly useful.—I. R. HARRIS.

KOENIGSBERGER, F. "*Welding Technology.*" Second edition. 8vo, pp. viii + 341. Illustrated. London, 1953 : Cleaver-Hume Press, Ltd. (Price 25*s*.).

The first edition of this book, published in 1949, gave prominence to the needs of the engineer responsible for the practical application of welding processes. It is clear that the author has sought to maintain this aim in the second edition, in which new material is, to a large extent, concerned with the important developments in welding technique during the past few years. Some of these developments have, partly of necessity, been dealt with briefly ; allowing for this, however, the book provides a unique coverage of present-day welding methods and equipment, and will serve the practising engineer as an extremely useful work of reference and information.

Processes brought into this edition include cold pressure welding, flame brazing of aluminium and its alloys, and twin-arc welding. In addition, an account is given of the further development of automatic fusion-welding methods. The welding of thermoplastic materials is described in a new chapter, which forms a useful primer on the subject. The author has also taken the opportunity of revising illustrations and references, and of bringing up to date various abstracts from standards and specifications. In all this the new or revised subject matter has, on the whole, been selected with care and good judgement, and only in the case of additions to the tables on butt- and fillet-welding procedure does the material appear less well chosen. Procedure for metal arc welding, in terms of electrode size, number of runs and welding current, is not a subject for rigorous standardization. As the author has indicated, there are several factors which would make it difficult, if not impossible, to specify exact and unalterable conditions for such procedure, even if this were desirable. The author's choice of tables, however, could mislead the reader in this respect, since some have been taken from a British Standard. Without reference to the fact that this Standard only purports to show the form of typical data sheets, as might be compiled by a drawing office, these tables could be mistakenly regarded as something more than the general guide which the author intended.

The chapters on ferrous and non-ferrous materials have been revised to present up-to-date recommendations for the application of various processes, notably argon arc welding, and nickel and its alloys have been added to the non-ferrous group. The author has not yet, however, brought stainless clad material under review—in some measure a surprising absentee in a book of this scope ; indeed the section on stainless steels generally, which remains substantially unaltered, might profitably have been re-cast to draw a more clear distinction between the advantages and limitations of welding processes in fabricating stainless-steel equipment. These points, however, cannot detract seriously from the value of a book which clearly contributes much to the understanding of the technology of welding and allied processes.—R. P. NEWMAN.

WHITEHOUSE, W. J., and J. L. PUTMAN. "*Radioactive Isotopes.*" An introduction to their preparation, measurement and use. 8vo, pp. xvi + 424. Illustrated. Oxford, 1953 : The Clarendon Press. (Price 50*s*.).

This book is described as an introduction to the preparation, measurement, and use of radioactive isotopes. It is written primarily for prospective users of these substances ; its contents will also appeal to the general reader who has some knowledge of physics. The early chapters provide a background of nuclear physics, dealing in turn with nuclear reactions of practical significance to isotope production, the more important modes of disintegration and the properties of the radiations. The theoretical and experimental facts are presented in a concise, clear and logical manner. A few sections involve mathematical expressions, otherwise this part of the book is a readable account of those aspects of radioactivity on which the applications of radioactive isotopes depend. A chapter is devoted to the production of radioactive isotopes in the uranium pile and in the cyclotron. The middle portion of the book is concerned with the measurements of the radiations and of their effects on matter, mainly by counting and ionization-chamber techniques. The description of the Geiger-Müller counter and its application to relative and absolute measurements of activity occupies the major portion of the longest chapter, which includes sections on scintillation counting and on the statistics of counting. Electronic circuits are not described. The so-called gross effects of the radiations (chiefly the production of ionization, with brief reference to calorimetry and chemical dosimetry) are dealt with in a separate chapter. The final portion, about one-third of the whole book, deals with the applications and the manipulation of radioactive material. The scope of applied radioactivity is now too wide for an overall survey in a book of this size, and the authors have accordingly classified the applications and given examples to illustrate the main methods. The numerous types of tracer experiment naturally receive the largest share of attention ; other topics discussed are the elimination of static electric charges, thickness measurements, and gamma-ray radiography. The plan adopted by the authors is well suited to meet the needs of the scientific worker in search of possible radioactive methods of investigation, and the general reader is provided with an admirable introduction to a rapidly growing subject. Of particular interest to the metallurgist are sections on self-diffusion, friction and lubrication, and microradiography. The precautions necessary to avoid contamination and hazards to health form the subject of the final chapter, in which it is emphasized that adequate safety in low-activity work can be ensured by quite straightforward procedures when these are rigidly adhered to. General and detailed references are appended to each chapter ; the information and references are based on material published before the end of 1950, with the addition of a few publications to which the authors had prior access. An abridged table of isotopes is included as an appendix.—W. E. PERRY.

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BRITISH STANDARDS INSTITUTION. British Standard A.104, June, 1953: "*Corrosion-Resisting Steel Bolts (Unified Hexagons and UNF Threads) for Aircraft.*" Folio, pp. 8. Illustrated. London, 1953: The Institution. (Price 3*s.* 6*d.*)

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BRITISH STANDARDS INSTITUTION. British Standard A.108, June, 1953. "*Steel Bolts (Unified Hexagons, UNF Threads and Close Tolerance Shanks) for Aircraft.*" Folio, pp. 7. Illustrated. London, 1953: The Institution. (Price 3*s.*)

BRITISH STANDARDS INSTITUTION. British Standard SP.53, June, 1953: "*Swaged Cable-End Assemblies (B.P. and B.S.F. Threads) for Preformed Steel Wire Rope for Aircraft (Suitable for Use with British Standard W.9 Wire Rope).*" Folio, pp. 10. Illustrated. London, 1953: The Institution. (Price 6*s.*)

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